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TRANSACTIONS

OF

THE FEDERATED INSTITUTION

OF

MINING ENGINEERS.

VOL. IX.—1894-95.

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BUTTERWORTH, GEORGE, Egstow, Market Street, Clay Cross, Chesterfield.
BUXTON, GERMAN, Reservoir Colliery, Moira, Ashby-de-la-Zouch.
BUXTON, SIMEON, Granville Colliery, Swadlincote, Burton-upon-Trent.
BYRNE, JNO. WALTER, Lawton Road, Alsager, Cheshire.

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CALVERT, A. F., The Mount, Oseney Crescent, Camden Road, London, N.W.
CAMERON, IAN, Sudbury, Ontario, Canada.
CAMPBELL, DOUGLAS, Tyack's Hotel, Camborne, Cornwall.
CAMPBELL, H. H., Sutton Hall, St. Helen's, Lancashire.
CAMPBELL, W. YOUNG, P.O. Box 357, Johannesburg, Transvaal.
CAMPBELL-JOHNSTON, R. C., P.O. Box 40, Vancouver, British Columbia.
CANDLER, T. E., East Lodge, Crook, by Darlington.
CAPELL, Rev. G. M., Passenham Rectory, Stony Stratford.
CARLOW, CHAS., The Fife Coal Company, Leven, Fife.
CARNES, CHARLES SPEARMAN, Cornwall House, Murton Colliery, Sunderland.
CARPENTER, S. W., 4, Tehidy Road, Camborne, Cornwall.
CARR, ARTHUR D., Whickham Rectory, Whickham, R.S.O.
CARR, THOMAS, Lilywhite Terrace, Four Lane Ends, Hetton-le-Hole, R.S.O., Co. Durham.
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CAZALET, L. P., Knight's Tribute Syndicate, Johannesburg, Transvaal.
CHALMERS, GEORGE, Superintendent of the St. John del Rey Mining Co., 28, Tower Chambers, Finsbury Pavement, London, E.C.
CHALMERS, J. A., P.O. Box 357, Johannesburg, Transvaal.
CHAMBERS, ALFRED, Eastwood, Notts.
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CHARLETON, A. G., Dashwood House, New Broad Street, London, E.C.
CHARLTON, W., Linares, Provincia de Jaen, Spain.
CHARLTON, WILLIAM, Alpine Villas, Bloxwich Road, Walsall.
CHATER, JOSEPH, Bengal Nagpur Coal Company, Limited, Evelyn Lodge, Kumarpur, Asansol, Bengal, India.
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CLIFFE, ALBERT, 23, Hardshaw Street, St. Helens.
CLIFFORD, E. H.
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CLOUGH, JAMES, Bomarsund House, Bomarsund, Bedlington, R.S.O.
CLOWES, Prof. F., University College, Nottingham.
COATS, WILLIAM A., Skelmorlie Castle, Skelmorlie, R.S.O., Ayrshire.
COBBOLD, C. H., Wentworth Castle, Barnsley.
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COCHRANE, W., St. John's Chambers, Grainger Street West, Newcastle-upon-Tyne.
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COHEN, E. H. A., P.O. Box 168, Barberton, Transvaal, South Africa.
COKE, G. E., 65, Station Street, Nottingham.
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COLLINS, ARTHUR LAUNCELOT, 14 and 15, Broad Street Avenue, London, E.C
COLLINS, HORATIO, P.O. Box 735, Johannesburg, Transvaal.
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COLQUHOUN, JAMES, 53, Flowerhill Street, Airdrie, N.B.
COLQUHOUN, T., 36, Sidney Grove, Newcastle-upon-Tyne.
COMMANS, R. E., 6, Queen Street Place, London, E.C.
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*DEAN, FRANK, Railway Foundry, Stoke-upon-Trent.
DEAN, SAMUEL W., 240, Waterloo Road, Burslem, Stoke-upon-Trent.
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*Dickinson, R. E., Bowling Iron Co., Limited, Bradford, Yorks. Transactions sent to Mrs. R. E. Dickinson, 2, Melrose Terrace, Rake Lane, Liscard, Cheshire.
Dixon, D. W., Lumpsey Mines, Brotton, R.S.O., Saltburn-by-the-Sea.
Dixon, James S., 97, Bath Street, Glasgow.
Dixon, Jonathan, Maitland Street, Stockton, *via* Newcastle, New South Wales.
Dixon, Walter, 164, St. Vincent Street, Glasgow.
Dobb, T. G., Derby Terrace, Hindley, near Wigan, Lancashire.
Dobbs, Joseph, Jarrow Colliery, Castlecomer, Co. Kilkenny.
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Dodds, A. P., 13, Dean Street, Newcastle-upon-Tyne.
Donald, A. W., 5, West Regent Street, Glasgow.
Donaldson, Robert, Clyde Iron Works, Tollcross, Glasgow.
Donkin, W., c/o A. M. Appleton, 10, Western Hill, Durham.
Dormand, Ralph B., Waldridge Colliery, Chester-le-Street.
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Douglas, John, 4, Brandling Terrace, Felling, R.S.O., Co. Durham.
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Drew, Walter Newton, Tankersley Cottage, Wharncliffe Silkstone, Barnsley.
Drinnan, Walter, Waverley Coal Company, Darnall, Sheffield.
Dron, R. W., 79, West Regent Street, Glasgow.
Dudley, Uriah, P.O. Box 110, Broken Hill, New South Wales.
Duignan, W. H., Gorway House, Walsall.
Dumat, Alfred, 1, Loop Street, Pietermaritzburg, Natal.
Dunbar, C., Wath Main Colliery, near Rotherham.
Duncan, C. Gregor, Collins Green Collieries, Earlestown, Lancashire.
Duncan, G. T., 110, Dilston Road, Newcastle-upon-Tyne.
Dundas, Edmund, Union Gold Mining Company, Limited, Barberton, Transvaal.
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Dunlop, Colin John, Quarter Collieries, Hamilton, N.B.
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Dunn, Robert, 178, Kenmure Street, Pollokshields, Glasgow.
Dunn, William L., 183, West George Street, Glasgow.
Dunsire, Andrew, 66, Main Street, Lochgelly, R.S.O., Fifeshire.
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EAMES, W., Rose Cottage, Midland Road, Nuneaton.
EARDLEY, EDWIN, 7, Wilson Street, Derby.
EARDLEY, J. W., The Grove, Alfreton.
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EASTON, ALEXANDER, 13, Southbridge Street, Bathgate, N.B.
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EMERSON, R., Tudhoe Colliery, Spennymoor.
EMINSON, J. B., Londonderry Offices, Seaham Harbour.
ENZOR, WILLIAM, Morton, Alfreton.
ETHERINGTON, J., 39A, King William Street, London Bridge, London, E.C.
EVANS, E. W., P.O. Box 357, Johannesburg, Transvaal.
EVANS, LEWIS, P.O. Box 601, Johannesburg, Transvaal.
EVANS, WILLIAM, Cliffe Vale, Stoke-upon-Trent.
EVERARD, J. B., 6, Millstone Lane, Leicester.

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FEARN, J. W., Devonshire Street, Chesterfield.
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FENWICK, BARNABAS, 84, Osborne Road, Newcastle-upon-Tyne.
FENWICK, P. J., The Sutton Colliery Company, Ltd., Sutton-in-Ashfield, Notts.
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FERENS, FRED. J., Silksworth Colliery, Sunderland.
FERENS, F. S., 13, Railway Arches, Westgate Road, Newcastle-upon-Tyne.
FERGIE, CHARLES, Drummond Colliery, Westville, Nova Scotia.
FERGUSON, D., c/o James Ferguson, 115 Buccleuch Street, Garnethill, Glasgow.
FERGUSON, KENNETH, Clay Cross Collieries, Chesterfield.
FERGUSON, PETER, 19, Royal Exchange Square, Glasgow.
FIELD, J., Hill Top, West Bromwich.
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FIRTH, WILLIAM, Water Lane, Leeds.
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FORD, JAS., Hotel Street, Coalville, near Leicester.
FORD, STANLEY H., c/o City and Suburban Gold Mining Company, P.O. Box 28, Johannesburg, Transvaal. All communications to c/o F. G. Ford, 19, North Street, Bristol.
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FORRESTER, ROBERT, Salt Lake City, Utah, U.S.A.
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FORSTER, T. E., 3, Eldon Square, Newcastle-upon-Tyne.
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FOULSTONE, W., Prince of Wales Hotel, Eldon Street North, Barnsley.
FOWLER, GEORGE, Basford Hall, Nottingham.
FOWLER, GEORGE C., Cliff Houses, Cinder Hill, Nottingham.
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French, John, Millcroft Cottage, Rutherglen Road, Glasgow.
Frew, Henry, Broomhouse Colliery, Broomhouse, N.B.
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Frew, James, 75, Bath Street, Glasgow.
Frew, Robert, 75, Bath Street, Glasgow.
Frew, William, Bredisholm Colliery, Baillieston, N.B.
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Gale, F. G., c/o H. W. Gale, Lincoln Bank, Brigg, Lincolnshire.
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Gallwey, A. Payne, P.O. Box 138, Johannesburg, Transvaal.
Gardiner, Edgar T., 8, South View Terrace, Bishop Auckland.
Gardner, Hugh, Minas Schwager, Coronel, Chili.
Garforth, W. E., West Riding Colliery, Normanton.
Garnett, Prof. Wm., 13, Spring Gardens, London, S.W.
Garside, F., Wath Main Colliery, near Rotherham.
Garven, William, 14, Lorne Terrace, Maryhill, Glasgow.
Gater, Enoch, Oak Tree Cottage, Talke, Stoke-upon-Trent.
Gavin, James, Udston Colliery, Hamilton, N.B.
Geddes, George H., 142, Princes Street, Edinburgh.
Gemmell, John, 10, St. Andrew's Square, Edinburgh.
Gemmell, Robert, Hopetown Works, Winchburgh, Linlithgowshire.
Gerrand, John, Crown Reef Gold Mining Company, Johannesburg, Transvaal.
Gerrard, James, 11, Meek's Buildings, Wigan.
Gerrard, John, H.M. Inspector of Mines, Worsley, Manchester.
Gibb, Robert, Glyncorwg, Cymmer, R.S.O., South Wales.
Gibbs, Hubert, Rookery Road, Handsworth, Birmingham.
Gifford, H. J., Minas de Passagem, Ouro Preto, Brazil.
Gilchrist, James, Lanemark Colliery, New Cumnock, R.S.O., Ayrshire.
Gilchrist, John, 2, Old Terrace, Muirkirk, R.S.O., Ayrshire.
Gilchrist, J. R., Garesfield Colliery, Lintz Green, Newcastle-upon-Tyne.
Gill, John, Lofthouse Colliery, Wakefield.
Gill, Thomas, Strafford Collieries, near Barnsley.
Gillespie, Alexander, Dalmellington Iron Works, Ayr, N.B.
Gillett, L. F., 163, Osmaston Road, Derby.
Gillman, F., Gartenstrasse, 1, Freiburg in/B., Baden, Germany.
Gillott, J. W., Summer Lane, Barnsley.
Gilmour, Allan, Portland Colliery, Kilmarnock, N.B.
Gilroy, S. B., Markham Collieries, Duckmanton.
Gjers, John, 3, Southfield Villas, Middlesbrough.
Gledhill, Edward, Carolina Hacienda, Honda, Republic of Colomba, South America.

GLEN, JOHN, Connell Terrace, Airdie, N.B.
GLEN, ROBERT, Eastfield Colliery, Cambuslang, Glasgow.
GLENNIE, W. H., 10, Oxford Road, Erdington, Birmingham.
GOERZ, A., P.O. Box 1961, Johannesburg, Transvaal.
GOLDIE, JAMES.
GOODALL, E. M., Eckington Collieries, Rotherham.
GOODMAN, Prof., Yorkshire College, Leeds.
GOODWIN, G. A., Roodeport, Witwatersrandt, South Africa.
GOODWIN, WILLIAM H., Chesterton Hall, Apedale, Staffordshire.
GOULDEN, Walter T., 3, Whitehall Place, London, S.W.
GORDON, GAVIN C., Netherton, Wishaw, N.B.
GOULDIE, JOSEPH, The North Eastern Bullfontine, Kimberley, South Africa.
GOWER, G. LEVESON, M.P., 14, South Audley Street, London.
GRAINGER, JAMES, Wollaton, Nottingham.
GRANT, A. M., Gardenhill, Kilmarnock, N.B.
GRANT, JOHN, The Grange, Larkhall, R.S.O., Lanarkshire.
GRATTON, R. T., Knifesmith Gate, Chesterfield.
GRAVE, L. W. DE, All Saints Works, Derby.
GRAVES, H. G., 5, Robert Street, Adelphi, London, W.C.
GRAY, ARCHIBALD, Middleton House, Dalry, Ayrshire.
GRAY, ARCHIBALD, Jun., 55, Eglinton Street, Ardrossan, N.B.
GRAY, JAMES, Riverside, Old Cumnock, N.B.
GRAY, JOHN, Morningside Colliery, Newmains, N.B.
GRAY, WILLIAM, Carfin Colliery (Dixon's), Motherwell, N.B.
GRAY, WILLIAM, Common Cottage, Auchinleck, R.S.O., Ayrshire.
GRAY, W. J., Birley Collieries, Sheffield.
GRAYSTON, F. A., White Lodge, Glascote, Tamworth.
GRAZEBROOK, A. W., Queen's Cross, Dudley, Worcestershire.
GREATBACH, GEORGE H., Great Fenton Collieries, Stoke-upon-Trent.
GREAVES, J. O., Westgate, Wakefield.
GREAVES, PERCY O., Westgate, Wakefield.
GREEN, Prof. A. H., 137, Woodstock Road, Oxford.
GREEN, A. T., Aldwarke Main Colliery, Rotherham.
GREEN, EDWIN R., Abbotsford Colliery, Abbotsford, Dunedin, New Zealand.
GREEN, J. T., Southside, Sneyd Park, Clifton, Bristol.
GREENE, THOS., Priors Lee, Shifnal. Shropshire.
GREENER, HENRY, South Pontop Colliery, Annfield Plain.
GREENER, T. Y., West Lodge, Crook, Darlington.
GREENSMITH, JOHNSON, Newstead Colliery, Nottingham.
GREENWELL, G. C., Jun., Poynton, near Stockport.
GREENWOOD, W. H., Portland Road, Edgbaston, Birmingham.
GREGORY, AUBREY, Dhadka Colliery, Asansol, E.I.R., India.
GREGORY, H. E., Cartonwood Colliery, Barnsley.
GREGORY, JOHN, Sneyd Colliery, Burslem, Stoke-upon-Trent.
GREGSON, G. E., 11, Chapel Street, Preston, Lancashire.
GRESLEY, W. S., Erie, Pennsylvania, U.S.A.
GREY, C. G., Ballycourcy, Enniscarthy, Co. Wexford.
GRIFFIN, GEORGE, Parkside, Market Street, Clay Cross, Chesterfield.
GRIFFITH, MAURICE, The Broughton and Plas Power Coal Co., Limited, near Wrexham.
GRIFFITH, N. R., Plasnewydd, Ruabon, North Wales.
GRIFFITHS, ED., Brymbo Colliery, Wrexham, North Wales.

GRIFFITHS, F., Pensnett, Dudley, Worcestershire.
GRIFFITHS, W., Waterloo House, Aberystwyth, South Wales.
GRIMSHAW, E. J., 23, Hardshaw Street, St. Helen's, Lancashire.
GRINHAFF, JOHN, Swadlincote Old Colliery, Burton-upon-Trent.
GUEST, ELI, Blackwell Collieries, Alfreton.
GURNEY, Rev. H. P., Durham College of Science, Newcastle-upon-Tyne.
GUTHRIE, JAMES K., Eltringham Colliery, Prudhoe, R.S.O.
GUTHRIE, REGINALD, Neville Hall, Newcastle-upon-Tyne.
GWYN, A. E. W., 114A, Queen Victoria Street, London, E.C.

HABERSHON, M. H., Thorncliffe Ironworks and Collieries, Sheffield.
HADDOCK, W. T., New Primrose Gold Mining Company, Limited, P.O. Box, 590, Johannesburg, Transvaal.
HADDOW, JOHN, Benquhat, Dalmellington, R.S.O., Ayrshire.
HADFIELD, JOSEPH, Holly Bank, Over Hulton, near Bolton, Lancashire.
HADFIELD, R. A., Hecla Steel Foundry, Sheffield.
HADLEY, WM., Shelton Steel and Iron Co., Stoke-upon-Trent.
HAGGIE, D. H., Wearmouth Patent Rope Works, Sunderland.
HAGGIE, F. W., Gateshead-upon-Tyne.
HAGGIE, G. A., Wearmouth Ropery, Sunderland.
HAGGIE, PETER SINCLAIR, Gateshead-upon-Tyne.
HAGUE, ERNEST, Castle Dyke, Sheffield.
HAINES, J. RICHARD, Adderley Green Collieries, Stoke-upon-Trent.
HALL, EDGAR, Wharncliffe Silkstone Colliery, Barnsley.
HALL, FRED., Fernleigh, Highfield, Workington.
HALL, FRED. W., Haswell Lodge, Sunderland.
HALL, GEORGE, Brassington Street, Clay Cross, Chesterfield.
HALL, JOHN, Sheepbridge Works, Chesterfield.
HALL, JOHN, Swadlincote and Cadley Hill Collieries, Burton-upon-Trent.
HALL, J. C., Pegswood Colliery, near Morpeth.
HALL, L. J., Furness Vale, Stockport.
HALL, M., Lofthouse Colliery, Wakefield.
HALL, M. S., 8, Victoria Street, Bishop Auckland.
HALL, R. O., Treeton, Rotherham.
HALL, TOM, Ryhope Colliery, *via* Sunderland
HALL, W. F., Haswell Colliery, Haswell, *via* Sunderland.
HALLAS, G. H., Wigan and Whiston Coal Co., Limited, Prescot.
HALLIMOND, W. T., P.O. Box 344, Johannesburg, Transvaal.
HALSE, EDWARD, Apartado, 512, Mexico, D.F.
HAMILTON, E., Rig Wood, Saltburn-by-the-Sea.
HAMILTON, G., c/o Messrs. B. S. Lloyd and Co., 78, Queen Victoria Street, London, E.C.
HAMILTON, JAMES, 208, St. Vincent Street, Glasgow.
HAMILTON, JOHN, 18, Orbiston Street, Motherwell, N.B.
HAMILTON, J. P., Bull Bridge Lime Works, Bull Bridge, Derbyshire.
HAMILTON, LORD, Dalzell Estate Office, Motherwell, N.B.
HAMILTON, ROBERT, 88, Great Clyde Street, Glasgow.
HAMILTON, WILLIAM, Niddrie Cottages, Portobello, N.B.
HANCOCK, H. R., Leetown, Horrabridge, Devonshire.
HANN, EDMUND, Aberaman, Aberdare.
HANNAH, DAVID, Ferndale, Glamorgan.

HARBIT, W. D., 32, High Street, Wallsend-upon-Tyne.
HARDIE, W. D. L., Birmingham, Alabama, U.S.A.
HARDMAN, JOHN E., 83, Hollis Street, Halifax, Nova Scotia.
HARDWICK, FRANCIS W., Firth College, Sheffield.
HARDWICK, FREDERICK, Eckington Collieries, Rotherham.
HARE, SAMUEL, Bedlington Collieries, Bedlington, R.S.O., Northumberland.
HARGREAVES, J., Rothwell Haigh Colliery, Leeds.
HARGREAVES, JOSEPH, Gwaun Cae Gurwen Collieries, Brynamman, R.S.O.
HARGREAVES, WALTER, Robin Hood Collieries, Wakefield.
HARGREAVES, WILLIAM, Rothwell Haigh Colliery, Leeds.
HARKER, WM., 17A, Great George Street, Westminster, London, S.W.
HARLE, PETER, Pagebank Colliery, Co. Durham.
HARLE, RICHARD, Browney Colliery, Durham.
HARLE, R. A., Hebburn Colliery, Hebburn-upon-Tyne.
HARP, RALPH, Bucknall, Stoke-upon-Trent.
HARPER, J. P., All Saints' Chambers, Derby.
HARRIS, G. E., Margherita, Debrugarh, Upper Assam.
HARRIS, W. S., Kibblesworth, Gateshead-upon-Tyne.
HARRISON, C. A., North-Eastern Railway, Newcastle-upon-Tyne.
HARRISON, GEORGE, High Park Colliery, Greasley, Nottingham.
HARRISON, G. B., H.M. Inspector of Mines, Swinton, near Manchester.
HARRISON, G. D., Waterworks, Hanley, Staffordshire.
HARRISON, W. B., Brownhills Collieries, near Walsall.
HARRUP, J. A., Westminster Collieries, Wrexham.
HART, ROBERT, Apedale, near Chesterton, Staffordshire.
HARVEY, W. PELLEW, Assay Office and Metallurgical Works, Vancouver, British Columbia.
HASSAM, ARTHUR, Waverley House, Fenton, Stoke-upon-Trent.
HASSAM, WILMOT J., Lane End Works, Fenton, Stoke-upon-Trent.
HASSARD, W. J., 209, Sauchiehall Street, Glasgow.
HASTIE, JAMES, Greenfield Colliery, Hamilton, N.B.
HASWELL COAL CO., Haswell Colliery, Haswell, *via* Sunderland.
HAWKINS, R. SPENCER, P.O., Salisbury, Mashonaland, South Africa.
HAWKINS, WALTER. Unstone, Sheffield.
HAY, J., Jun., Widdrington Colliery, Acklington.
HAY, T. Y., Whitwick Colliery, Coalville, Leicester.
HAY, WILLIAM, Nostell Colliery, Wakefield.
HAY, W., Stanton Colliery, Burton-upon-Trent.
HAYWARD, W. J., West Bromwich.
HEAD, JEREMIAH, 47, Victoria Street, Westminster, London, S.W.
HEAP, R. S., 2, Nelson Square, Burnley.
HEATH, A. S., Adderley Green Collieries, Stoke-upon-Trent.
HEATH, JAMES, Clayton Hall, Newcastle, Staffordshire.
HEATH, JOHN, Sneyd Colliery, Burslem, Stoke-upon-Trent.
HEATH, ROBERT, Biddulph Valley Iron Works, Stoke-upon-Trent.
HEATH, WILLIAM, Sneyd House, Burslem, Stoke-upon-Trent.
HEATHCOTE, C. H., Newstead Colliery, Nottingham.
HEDLEY, A. M., Medomsley, R.S.O., Co. Durham.
HEDLEY, E., Rainham Lodge, The Avenue, Beckenham, Kent.
HEDLEY, J. HUNT, John Street, Sunderland.
HEDLEY, J. L., H.M. Inspector of Mines, 2, Devonshire Terrace, Newcastle-upon-Tyne.

HEDLEY, SEPT. H., Langholme, Roker, near Sunderland.
HEDLEY, W. H., Medomsley, R.S.O., Co. Durham.
HENDERSON, C. W. C., The Riding, Hexham.
HENDERSON, H., Pelton Colliery, Chester-le-Street.
HENDERSON, J. J., Khewra, Sind Sagar S. Railway, Punjab, India.
HENDERSON, J. J., U.S. Engineer's Office, Kingsbridge, New York, U.S.A.
HENDERSON, WILLIAM, Levenseat, Fauldhouse, Linlithgowshire.
HENDERSON, WILLIAM, Wheatley Hill Colliery, *via* Trimdon Grange.
HENDY, J. C. B., Etherley, *via* Darlington.
HENSHAW, A. MAYON, Talk-o'-th'-Hill Colliery, Talke, Stoke-upon-Trent.
HENSHAW, JOHN, Butterley Park, Alfreton.
HENZELL, ROBERT, Close, Newcastle-upon-Tyne.
HEPBURN, T., The Firs, Langley Park, Durham.
HEPPELL, T., Leafield House, Birtley, Chester-le-Street.
HEPPLEWHITE, W. H., H.M. Inspector of Mines, Roscius House, Corporation Oaks, Nottingham.
HERD, THOMAS, Dunnikier Colliery, Kirkcaldy, N.B.
HERSCHEL, Prof. A. S., Observatory House, Slough, Bucks.
HESLINGTON, ALFRED, New Tupton, Chesterfield.
HESLOP, C., Woodside, Marske Mill Lane, Saltburn-by-the-Sea.
HESLOP, GRAINGER, Deptford Hall, Sunderland.
HESLOP, JAMES, Ashfield, Gedling, near Nottingham.
HESLOP, MICHAEL, Rough Lea Colliery, Willington, Co. Durham.
HESLOP, THOMAS, Storey Lodge Colliery, Cockfield, *via* Darlington.
HETHERINGTON, J. RIDLEY, c/o Mrs. Burnley, Sisters Villas, Garforth Colliery, near Leeds.
HETTON COAL COMPANY, Fence Houses.
HEWITT, C. R., London Road, Derby.
HEWITT, G. C., Serridge House, Coal Pit Heath, near Bristol.
HEWITT, H. R., H.M. Inspector of Mines, 23, Hartington Street, Derby.
HEWITT, JOSEPH, Heron Cross, Fenton, Stoke-upon-Trent.
HEWLETT, A., Haseley Manor, Warwick.
HICKMAN, E., Stone Quarries, Bilston.
HICKS, JAMES, Redruth, Cornwall.
HICKS, Prof. W. M., Firth College, Sheffield.
HIGBY, ROBERT GEORGE, Borrea Coal Company, Limited, Sitarampore, Bengal, East India Railway.
HIGGINBOTTOM, REGINALD, Sheepbridge Works, Chesterfield.
HIGGINS, SAMUEL, Weston Coyney Road, near Longton, Staffordshire.
HIGSON, JACOB, Crown Buildings, 18, Booth Street, Manchester.
HIGSON, JOHN, Crown Buildings, 18, Booth Street, Manchester.
HILL, THOMAS, Codnor Park, Alfreton.
HILL, WILLIAM, Manor Street Foundry, Fenton, Stoke-upon-Trent.
HILL, WILLIAM, Old Schoolhouse, Dalmellington, R.S.O., Ayrshire.
HILL, WILLIAM, The Pooley Hall Colliery Company, Polesworth, near Tamworth.
HILTON, J., 67, Hawkshead Street, Southport.
HILTON, T. W., Wigan Coal and Iron Co., Limited Wigan.
HINCHLIFFE, J., Bullhouse Colliery, Penistone, near Sheffield.
HIRST, G. F., Rotherham Main Colliery, near Rotherham.
HOBBS, WILLIAM L., Whitfield Collieries, Norton-in-the-Moors, Stoke-upon-Trent.
HOBSON, MOSES, Shildon, *via* Darlington.
HODGE, FRANCIS, Coromandel, near Auckland, New Zealand.

HODGES, ISAAC, Blackwell Collieries, Alfreton.
HODGKINSON, ALBERT, The Laurels, Wollaton, Nottingham.
HODGSON, E., Black Boy Colliery, Bishop Auckland.
HODGSON, J., Edmondsley Colliery, Chester-le-Street.
HODGSON, L H., Garforth Colliery, Leeds.
HODSON, JAMES, The Mount, Thalto Heath, St. Helen's, Lancashire.
HODSON, JAMES E., Belgrave Terrace, Normacot, Stoke-upon-Trent.
HOGG, JAMES, Kirkhill Colliery, Cambuslang, Glasgow.
HOGG, JOHN, Ardoch Grove, Cambuslang, Glasgow.
HOGG, JOHN, Victoria Engine Works, Airdrie, N.B.
HOLBROOK, JOHN, Langley Colliery, Marlpool, Derby.
HOLDING, WILLIAM, Cossall Colliery, Nottingham.
HOLFORD, W. D., Whittington, Chesterfield.
HOLIDAY, ROSLYN, Ackton Hall Colliery, Featherstone, near Pontefract.
HOLLAND, WALTER, Brampton Colliery, Chesterfield.
HOLLIDAY, MARTIN F., Langley Grove, Durham.
HOLLIDAY, T., West Ardsley Collieries, Tingley, near Wakefield.
HOLLINGS, JOHN H., Wath Main Colliery, Rotherham.
HOLLINS, G., High Street, Wolstanton, Stoke-upon-Trent.
HOLLIS, HENRY WM., Whitworth House, Spennymoor.
HOLLIS, ROBERT, Ansley, Atherstone.
HOLMAN, J. M., Hill Crest, Camborne, Cornwall.
HOLMES, C., 109, Victoria Road, Darlington.
HOLT, JOHN, Jun., 109, Manchester Road, Swinton, Lancashire.
HOMER, J. EDWARD, Ivy House, Hanley, Staffordshire.
HOOD, A., 6, Bute Crescent, Cardiff.
HOOD, FREDERICK, 24, Linton Heath, Burton-upon-Trent.
HOOD, JAMES A., Rosewell, Midlothian.
HOOPER, ED., c/o J. H. Hooper, College Precincts, Worcester.
HOOPER, G. C., Whitehaven Collieries, Whitehaven.
HOPKINS, EDWARD, 13, Harrington Gardens, London, S.W.
HOPKINSON, HENRY, Station Street, Nottingham.
HOPKINSON, JOHN, Inglewood, St. Margaret's Road, Bowden, Cheshire.
HOPPER, J. I., Wire Rope Works, Thornaby-upon-Tees.
HORSFIELD, A., Horsbury Junction Iron Works, Wakefield.
HOSKING, JAMES, Roskear, Camborne, Cornwall.
HOSKING, W., Roskear Villas, Camborne.
HOSKOLD, H. D., Inspector-General of Mines of the Argentine Republic, and Director of the National Department of Mines and Geology, Casilla Correos, 900, Buenos Ayres.
HOUFTON, CHARLES, Heathfield Villas, Garforth, Leeds.
HOUFTON, J. P., Bolsover Colliery, Chesterfield.
HOULGATE, J. KERR, 69, Lowther Street, Whitehaven.
HOUSTON, JAMES, Salsburgh, Holytown, R.S.O., Lanarkshire.
HOWARD, W. F., 15, Cavendish Street, Chesterfield.
HOWAT, J. T., Eglinton Iron Works, Killwinning, R.S.O., Ayrshire.
HOWAT, ROBERT M., 32, Victoria Street, Westminster, London, S.W.
HOWAT, WILLIAM, Rankinston Works, Ayr, N.B.
HOWDEN, THOMAS, Wakefield.
HOWE, WM., Clay Cross, Chesterfield.
HOWELL, J. C., New Dock, Llanelly, South Wales.
HOWES, FRANK T., Singareni Collieries, Hyderabad Deccan Co., Secunderabad, India.

Howie, John, Machan Colliery, Larkhall, R.S.O., Lanarkshire.
Howieson, Robert, Moss End, West Calder, R.S.O., Midlothian.
Howl, E., The Quarries, Dudley, Worcestershire.
Hubbersty, H. A., Burbage, Buxton.
Hugh, James, Thornhill, Blantyre, Glasgow.
Hughes, H. W., Priory Farm House, Dudley, Worcestershire.
Hughes, James, Brykinalt Colliery, Chirk, North Wales.
Hughes, J., Dudley, Worcestershire.
Hughes, T. Vaughan, Chemical and Physical Laboratory, Greenfield, Holywell, near Chester.
Hughes, William, Springwood, Chesterton, Staffordshire.
Hulett, Charles T., Lilleshall Collieries, St. Georges, Wellington, Salop.
Hull, Prof. Edward, 20, Arundel Gardens, London, W.
Humble, Joseph, Markham Collieries, Duckmanton, Chesterfield.
Humble, Stephen, Jun., 9, Victoria Street, Westminster, London, S.W.
Humble, William, Coal-fields Office, Newcastle, New South Wales.
Humphreys-Davies, G., 8, Laurence Pountney Hill, Cannon Street, London, E.C.
Hunt, Walford, Fife Street, Nuneaton.
Hunter, Andrew, Alloa Colliery, Alloa, N.B.
Hunter, Christopher, Broomhill Colliery, Acklington, Northumberland.
Hunter, David, 101, St. Vincent Street, Glasgow.
Hurll, Mark, 95, Bath Street, Glasgow.
Hurst, George, 58, Eldon Street, Newcastle-upon-Tyne.
Hurst, G. A., Dosthill, Tamworth.
Hutton Henry Coal Company, Limited, 7, Bondgate, Darlington.
Hyslop, G. P., Mossfield Colliery, Longton, Staffordshire.
Hyslop, Jonathan, Airdriehouse Cottage, Airdrie, N.B.

I'Anson, James, Fairfield House, Darlington.
Imeson, J. A., Wearmouth Colliery, Sunderland.
Ingold, Herbert, c/o W. Cooke & Co., Limited, Tinsley Steel and Iron Works, Sheffield.
Irvine, John, Bothwell Castle Colliery, Bothwell, N.B.
Isherwood, Jonathan, Hoodroyd Colliery, Barnsley.

Jackson, Alfred, Blythe Marsh, Stoke-upon-Trent.
Jackson, David, Glengyron, Old Cumnock, N.B.
Jackson, Douglas, Coltness Iron Works, Newmains, N.B.
Jackson, H. H., Colley Gate, Cradley, Staffordshire.
Jackson, John, Stubben Edge, Chesterfield.
Jackson, W. B. M., Clay Cross Hall, Chesterfield.
Jackson, W. G., Hicklam House, Aberford, near Leeds.
James, A. A., Croxdale, near Durham.
James, H. M., Colliery Office, Whitehaven.
Jamieson, Alexander, Balgonie Colliery, Markinch, N.B.
Jamieson, J. Auldjo, 66, Queen Street, Edinburgh.
Jamieson, T. J., Eddlewood Colliery, Hamilton, N.B.
Janson, E. W., 7 and 9, Leadenhall Buildings, London, E.C.
Jarratt, J., Houghton Main Colliery, Barnsley.
Jeffcock, C. E., Birley Collieries, Sheffield.
Jeffcock, T. W., 18, Bank Street, Sheffield.
Jeffrey, William, Ladylands Colliery, Shotts, R.S.O., Lanarkshire.

JENKINS, W., Ocean Collieries, Treorky, Glamorganshire.
*JENKINS, WM., Consett Iron Works, Consett, Durham.
JEPSON, H., 20, The Avenue, Durham.
JEPSON, W. W., Portland Colliery, Selston, Alfreton.
JEUDWINE, W. W., Walton Lodge, Chesterfield.
JOBLING, T. E., Croft Villa, Blyth, Northumberland.
JOHNS, J. HARRY, P.O. Box 1021, Johannesburg, Transvaal.
JOHNSON, GUY ROCHE, Embreville, Washington County, Tennessee, U.S.A.
JOHNSON, H., Blackett Colliery, Haltwhistle.
JOHNSON, HENRY, Trindle Road, Dudley, Worcestershire.
JOHNSON, J., The Terrace, Stairfoot, near Barnsley, and Charbonnages de Rodez, Limited, Gages, Avéyron, France.
JOHNSON, WM., Framwellgate Moor, Durham.
JOHNSON, WM., Radcliffe Colliery, Acklington, Northumberland.
JOHNSTON, J. HOWARD, c/o Messrs. Backus and Johnston, Lima, Peru, South America.
JOHNSTONE, HUGH, 77, Montgomery Street, Kelvinside North, Glasgow.
JOHNSTONE, JAMES, Belhaven Road, Wishaw, N.B.
JOHNSTONE, RONALD H., 5, West Regent Street, Glasgow.
JOICEY, J. G., Forth Banks West Factory, Newcastle-upon-Tyne.
JOICEY, JAMES JOHN, Sunningdale Park, Berks.
JOICEY, W. J., Sunningdale Park, Berks.
JONAS, GEO. B., 15, Gordon Square, Glasgow.
JONES, D. L., Shelton Steel Co., Stoke-upon-Trent.
JONES, ENOS, Whitwell Colliery, Whitwell, Chesterfield.
JONES, F. J., Rother Vale Collieries, Treeton, Rotherham.
JONES, ISAAC, Wynnstay Collieries, Ruabon.
JONES, JOHN, Shire Oaks Colliery, Worksop.
JONES, J. A., Gijon, Asturias, Spain.
JONES, JACOB CARLOS, Bellambi, New South Wales.
JONES, LLOYD, Ruabon, North Wales.
JONES, THOMAS, Netherton, near Dudley, Worcestershire.
JONES, THOMAS E., Shelton Bar Iron Works, Stoke-upon-Trent.
JONES, WILLIAM, Trimdon Grange Colliery, Trimdon Grange, R.S.O.

KANDA, REIJI, c/o Mr. Takahira Kanda, 9, Awajiche-Nichome, Kanda, Tokio, Japan.
KAY, ROBERT, South Tanfield, Stanley, R.S.O., Co. Durham.
KAYLL, A. C., Gosforth, Newcastle-upon-Tyne.
KAYSER, H. W. FERD., The Mount Bischoff Tin Mining Co., Waratah, Tasmania.
KEIRS, JOHN, Drongan Castle Colliery, Drongan, Ayr, N.B.
KELL, G. J., Beech Grove, Locke Park, Barnsley.
KELLET, M. H., 5, Elm Street, South Moor, Chester-le-Street.
KELLETT, WILLIAM, Portland Bank, Southport.
KENDALL, J. D., Foxhouses Road, Whitehaven.
KENNEDY, HENRY, Fauldhouse, N.B.
KENNEDY, RANKIN, The Phœnix Dynamo Manufacturing Co., Hubert Street, Leeds Road, Bradford.
KENNEDY, WILLIAM, Gilburn Cottage, Shotts, N.B.
KENRICK, JOHN P., 8, St. James' Square, Manchester.
KENT, GEORGE C., Longton, Staffordshire.
KENYON, G. C., Sandygate, Wath-upon-Dearne, Rotherham.

KERR, ARCHIBALD, Mayville, Cleland, Lanarkshire, N.B.
KESTEVEN, FRANK, Monckton Main Colliery, Barnsley.
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SIBOLD, C. W., Public Works Department, Soane Circle, Arrah, Bengal, India.
SIDEBOTHAM, J. N., 1, Princess Street, Albert Square, Manchester.
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SQUIRE, J. B., 297, Clapham Road, London, S.W.
STAFFORD, THOMAS W., 1, Lea Road, Heaton Chapel, Stockport.
STALEY, A. H., 21, Coton Road, Nuneaton.
STANDLEY, WILLIAM, Nuneaton.
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STOBART, W., Pepper Arden, Northallerton.
STOBBS, J. T., Walker Colliery, Walker-upon-Tyne.
STOKER, ARTHUR P., Birtley, near Chester-le-Street.
STOKER, HENRY, 5, Argyll Mount, Mansfield.
STOKER, THOMAS E., 37, Whitehouse Lane, Ushaw Moor, Durham.
STOKES, JOHN, Tollcross Colliery, Tollcross, Glasgow.
STOKOE, JAMES, 7, North Street, Murton Colliery, *via* Sunderland.
STONES, GEORGE B., Dale View, Conisbro', Rotherham.
STOREY, THOS. E., Weston Coyney Hall, Longton, Staffordshire.
STRACHAN, ALEXANDER, 131, Station Street, Burton-upon-Trent.
STRAIN, HUGH, Cleddans, Easterhouse, Baillieston, Glasgow.
STRAIN, JAMES M., 11, Hillend Gardens, Patrickhill, Glasgow.
STRAKER, J. H., Howden Dene, Corbridge-upon-Tyne.
STRATTON, H. S., Cramlington Collieries, Northumberland.
STREATFIELD, HUGH S., Ryhope, near Sunderland.
STREET, JOHN, High Street, Silverdale, Staffordshire.
STRICK, JOHN, Bar Hill, Madeley, Staffordshire.
STRICK, R. J., Yniswern, Mumbles, Swansea.
STRICK, THOMAS SHEPHARD, Bar Hill, Madeley, Staffordshire.
STROUD, Prof. H., Durham College of Science, Newcastle-upon-Tyne.
STUART, C. M., St. Dunstan's College, Lewisham.
STUART, DONALD M. D., Redland, Bristol.
STUBBS, THOMAS, Aldwarke Main Colliery, Rotherham.
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SWALLOW, J., Catchgate, Annfield Plain, R.S.O., Co. Durham.
SWALLOW, J. F., Mosboro' Hill House, Eckington, Rotherham.
SWALLOW, R. T., Simonside House, Simonside, South Shields.
SWALLOW, W. A., Bushblades House, Lintz Green.
SWAN, H. F., North Jesmond, Newcastle-upon-Tyne.
SWAN, J. G., Upsall Hall, near Middlesbrough.
SWANN, FRANK, Annesley, Nottingham.
SYKES, FRANK K., Springwell Villa, Bishop Auckland.

TAIT, JAMES, South Kirkby Colliery, near Wakefield.
TAIT, JOHN R., 204, Morrison Street, Edinburgh.
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TATE, SIMON, Trimdon Grange Colliery, Co. Durham.
TATE, WILLIAM, Measham Main Colliery, Measham, near Atherstone.
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TAYLOR, T., Quay, Newcastle-upon-Tyne.
TAYLOR, WILLIAM, Tibberton Grange, Newport, Salop.
TAYLOR-SMITH, THOMAS, Broadwood Park, Lanchester, Co. Durham.
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THOMAS, R., Bloxwich, Walsall.
THOMAS, W., Penelvan, Camborne, Cornwall.
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THOMPSON, EDWARD, Norton Collieries, Stoke-upon-Trent.
THOMPSON, EDWARD, 70, Queen's Walk, Nottingham.
*THOMPSON, F. A., Broad Street House, London, E.C.
THOMPSON, JOSEPH, 38, Church Street, Murton Colliery, Sunderland.
THOMPSON, J. W., Backworth Colliery, Newcastle-upon-Tyne.
THOMPSON, MATTHEW E., Chatterley Whitfield Collieries, Bucknall, Stoke-upon-Trent.
THOMPSON, R., Jun., 19, The Crescent, Gateshead-upon-Tyne.
THOMPSON, THOMAS, Featherstone Main Collieries, Pontefract.
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THOMSON, JOHN B., Fairview House, Hamilton, N.B.
THOMSON, JOS. F., Manvers Main Colliery, Wath-upon-Dearne, Rotherham.
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TREGLOWN, C. H., 38, Wretham Road, Handsworth, Birmingham.
TRELEASE, W. HENWOOD, Pestarena, Vall Anzasca, Novara, Italy.
TRESTRAIL, N., Claremont Road, Redruth.
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WARBURTON, J. S., 19, Stanwick Road, West Kensington, London, W.
WARD, A. H., c/o Bengal Coal Company, Limited, Calcutta, India.
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WATSON, WILLIAM, 41, Ann Street, Glasgow.
WATSON, WILLIAM, Livingstone Terrace, Uddingston, Glasgow.
WATTS, JOHN, Edensor Street, Chesterton, Staffordshire.
WEARMOUTH COLLIERY, OWNERS OF, Sunderland.
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WHITE, H., Walker Colliery, Newcastle-upon-Tyne.
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WHITEHEAD, JOHN JAMES, 85, Haworth Buildings, Cross Street, Manchester.
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WHITELAW, JOHN, 118, George Street, Edinburgh.
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WILLIAMS, EDMUND W., Henshall House, Goldenhill, Stoke-upon-Trent.
WILLIAMS, ERNEST, P.O. Box 965, Bittelheim Buildings, Simmonds Street, Johannesburg, Transvaal.

WILLIAMS, HERBERT IGNATIUS, Goldenhill, Stoke-upon-Trent.
WILLIAMS, J., 115, Wellington Road, Dudley, Worcestershire.
WILLIAMS, THOMAS, Goldthorn Hill, Wolverhampton.
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WILSON, ROBERT, Bathville Collieries, Armadale, N.B.
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WINCHELL, HORACE V., 1306, S.E. 7th Street, Minneapolis, Minn., U.S.A.
WINGATE, JOHN B., 208, St. Vincent Street, Glasgow.
WINSTANLEY, PETER, Shaw Cross Colliery, near Dewsbury.
WINSTANLEY, ROBT., 28, Deansgate, Manchester.
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WOOD, T. P., Market Place, Chesterfield.
WOOD, W. H., Coxhoe Hall, Coxhoe, R.S.O., Co. Durham.
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WOOD, WALTER K., Broomhouse Colliery, Broomhouse, Glasgow.
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WOODHEAD, L., Beeston Colliery, Leeds.
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WOOLCOCK, J. H., 49, Lowther Street, Whitehaven.
*WOOLLISCROFT, FREDERICK, St. Peter's Chambers. Stoke-upon-Trent.
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YATES, JOHN, 52, Lower Broughton Road, Lower Broughton, Manchester.
YEOMAN, T. P., Manager, Bengal Coal Company's Collieries, Laikdih, E.I.R., Bengal, India.
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ZELLER, C. VAN, Basset Street, Camborne, Cornwall.
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HERD, SAMUEL, Balgonie, Markinch, N.B.
HILL, JAMES, Helenbank Cottage, Dalmellington, R.S.O., Ayrshire.
HILLHOUSE, WILLIAM, 61, Fullarton Street, Kilmarnock, N.B.
HOGG, WILLIAM, Carnmoor Cottage, Forth, Lanark, N.B.
HOOD, ARCHIBALD, 6, Bute Crescent, Cardiff.
HOWAT, WILLIAM, Sundrum, Coylton, Ayr, N.B.
HUSBAND, JOHN, Tannochside, Uddingston, Glasgow.
HYSLOP, WILLIAM, Bank Colliery, New Cumnock, R.S.O., Ayrshire.

KELSO, JOHN, Springfield Place, High Blantyre, Glasgow.
KESSON, ANDREW, Clydebank House, Bothwell, Glasgow.
KING, PETER, Darngavil, Airdrie, N.B.

LANDALE, Dr. DAVID, The Binn, Burntisland, N.B.
LOGAN, WILLIAM, Almond House, Linlithgow, N.B.
LONGBOTTOM, L., Stoke-upon-Trent.
LOVE, JOHN, Shottstown, Penicuik, N.B.
LUCAS, JAMES, The Limes, Lawton, Staffordshire.

M'CANN, EDWARD, 9, Westpans, Musselburgh, N.B.
M'CULLOCH, JOHN, Jawcraig Colliery, Slamannan, Stirlingshire.
M'CULLOCH, JOHN, Loganlea Colliery, West-Calder, R.S.O., Midlothian.
M'DONALD, JOHN, Greenhill Colliery, Holytown, R.S.O., Lanarkshire.
M'GEE, JOHN, Tillicoultry Colliery, Tillicoultry, R.S.O., Clackmannanshire
M'GILL, JAMES, Burntisland Oil Works, Burntisland, N.B.

M'KENZIE, WILLIAM, Lothian Street, Rosewell, Edinburgh.
M'KILLOP, JAMES, Polmont Park, Polmont, Stirlingshire.
M'KINNELL, WILLIAM, 30, South Grove Road, Sheffield.
MARTIN, DANIEL, Clydeside, Broomhouse, Glasgow.
MAXWELL, JAMES, Limerigg Cottage, Slamannan, Stirlingshire.
MAXWELL, PETER, Redding Muirhead, Polmont Station, Stirlingshire.
MORISON, JOHN, Cramlington House, Northumberland.
MORROW, SAMUEL, Palacecraig, Airdrie, N.B.

NASMYTH, A., Donibristle Colliery, Crossgates, Fife.
NASMYTH, J. A., Donibristle Colliery, Crossgates, Fife.
NEILSON, BENJAMIN, Prestongrange Coal Co., Prestonpans, R.S.O., Haddingtonshire.
NIMMO, THOMAS, Kinneil Colliery, Bo'ness, R.S.O., Linlithgowshire.

OSBORNE, CHARLES, 72, Piccadilly, Hanley, Staffordshire.

PLATT, SAMUEL R., Oldham.

RANKIN, GEORGE, Victoria Works, Nitshill, Hurlet, Glasgow.
RENNIE, WILSON, Gateside Colliery, Cambuslang, Glasgow.
ROBERTSON, DAVID, 135, Waterloo Street, Glasgow.
ROBERTSON, Dr. J. R. M., Linton, Wilson's Point, Sydney, New South Wales.
ROBERTSON, JOHN, Jun., 12, St. Vincent Place, Glasgow.
ROBERTSON, ROBERT, Jun., Sydney Mines, Cape Breton, Nova Scotia.
RUSSELL, ARCHIBALD, Jun., Avongrange, Hamilton, N.B.
RUSSELL, JOSEPH, Westburn Colliery, Newton, Glasgow.
RUSSELL, P. B., Newfield, Hamilton, N.B.
RUSSELL, ROBERT, Coltness Iron Works, Newmains, R.S.O., Lanarkshire.

SCOTT, WILLIAM, 56, Braid Road, Edinburgh.
SHARP, ROBERT, Coatbridge, N.B.
SHAW, JOHN, Ferniegair, Hamilton, N.B.
SMITH, EUSTACE, Jun., St. Lawrence Ropery, Newcastle-upon-Tyne.
SMITH, JOHN, Cannock Lodge Colliery, Bloxwich, Staffordshire.
SMITH, JOHN, Rosehall Colliery, Coatbridge, N.B.
SPARROW, WM. B., 39, Clydesdale, Wishaw, N.B.
SPEIR, WILLIAM, Balgonie Colliery, Thornton, R.S.O., Fifeshire.
STIRLING, JOHN, Bonnybridge, Stirlingshire.
STRACHAN, PETER, 94, Aitchieson Street, Airdrie, N.B.

TREGLOWN, C. H., 38. Wretham Road, Handsworth, Birmingham.
TWADDLE, WILLIAM, Ferniegair, Hamilton, N.B.

WADDELL, ALEXANDER W., Glenburn, Uddingston, Glasgow.
WARDROPE, JAMES, Fordell Colliery, Crossgates, Fife.
WATT, ROBERT, Brownlee Colliery, Law, Carluke, N.B.
WEIR, THOMAS D., Ferro Carril Central de Venezuela, Caracas, Venezuela.
WELSH, WILLIAM, Craigmark, Dalmellington, R.S.O., Ayrshire.
WHITE ANDREW, Ormiston Colliery, Ormiston, Hadddingtonshire.
WHITE, WILLIAM, 46, Bothwell Square, Bothwell, N.B.
WHITTON, JOSEPH, 93, Flemington, Cambuslang, Glasgow.
WOODALL, WILLIAM, Bleak House, Burslem, Stoke-upon-Trent.
WYLIE, JOHN, Clifton Iron Works, Coatbridge, N.B.

YOUNG, WILLIAM, 204, Stobcross Street, Glasgow.

TRANSACTIONS

OF

THE FEDERATED INSTITUTION

OF

MINING ENGINEERS.

THE MINING INSTITUTE OF SCOTLAND.

GENERAL MEETING,

HELD IN THE ROOMS OF THE CHRISTIAN INSTITUTE, GLASGOW, FEBRUARY 6TH, 1894.

MR. GEORGE A. MITCHELL, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and confirmed.

Office-bearers for the session 1895-96, were nominated.

The following gentlemen were elected:—

FEDERATED MEMBERS—

Mr. WILLIAM W. SOMERVILLE, 1, Peden's Place, Benhar, Whitburn, N.B.
Mr. WALTER H. MUNGALL, Transy, Dunfermline.

The following paper was read by Mr. ALLAN STIRLING on "Water-tube Boilers":—

WATER-TUBE BOILERS.

By ALLAN STIRLING.

Previous to the year 1880, water-tube boilers were very little used, but since that time their use has been gradually extending, until at the present time there are probably more boilers of that type built than on the fire-tube system. There were several reasons for this change from fire-tube to water-tube boilers, but chiefly because a properly designed water-tube boiler is the most safe, economical, efficient, and durable.

The first meeting of the American Society of Mechanical Engineers was held in 1880, and the writer contributed a paper on the subject of "Practical Methods for the Economy of Steam."* The use of higher pressures was strongly advocated in that paper as one of the elements of economy. Pressures as high as 300 pounds per square inch were suggested, and a statement made as to the economy of using steam at that pressure as compared with steam at the then common pressure of 75 pounds per square inch.

In considering the question of the most efficient boiler at that time, the water-tube system was not even mentioned by the writer, and drawings were given of a special construction of a fire-tube boiler suitable for the production of high-pressure steam.

In the year 1885, the writer read another paper before the same society on the subject of "Shell and Water-tube Boilers,"† showing that the water-tube system had made headway during the interval between the two papers being written. In preparing the latter paper, occasion was taken to go into the relative merits of the two systems, and the writer became convinced that the water-tube system had great advantages over the older type. The only types of water-tube boiler at that time in the market were the Babcock and Wilcox, the Root, the Heinie, etc.; these types consist of a series of tubes slightly inclined from the horizontal, expanded at both ends into headers, and connected to the steam-and-water reservoir above and to a mud-drum below.

In 1886, the writer constructed a water-tube boiler, which was worked at a pressure of 200 pounds per square inch. It was composed entirely

* *Proceedings American Society of Mechanical Engineers*, vol. i., page 18.

† *Ibid.*, vol. vi., page 566.

of Field tubes, screwed into the bottom plate of a rectangular steam-and-water receiver. This boiler was worked for a short time, but the sediment gradually accumulated at the lower end of the tubes, and the boiler was ultimately thrown upon the scrap-heap.

In 1887, another boiler similar to the former was built, with the addition of a mud-drum, to which two rows of tubes were connected, the idea being that the circulation of the water produced by means of the tubes through the mud-drum would prevent the sediment from collecting in the bottom ends of the Field tubes. This arrangement proved to be only partially successful, but this boiler is still doing good work.

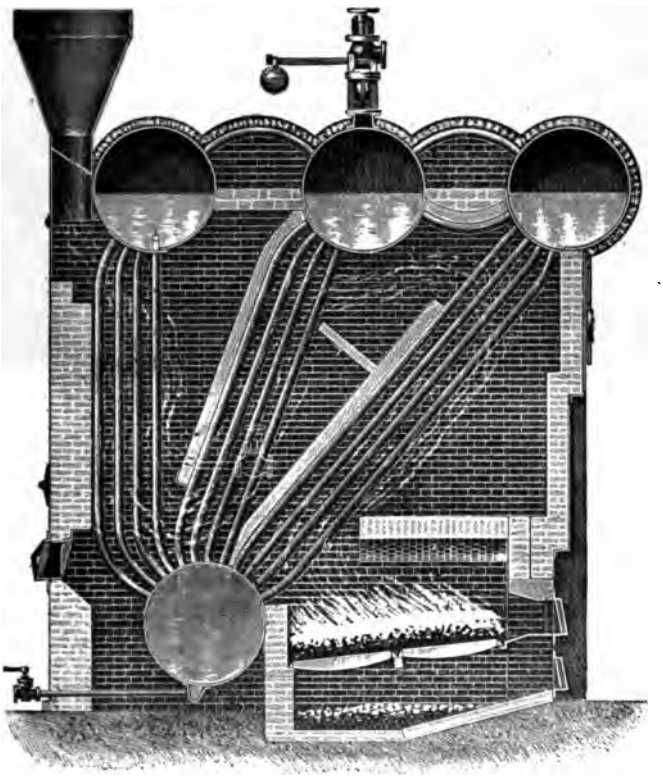


Fig. 1.

The writer then decided to discard the Field tubes, and used two cylinders connected by tubes instead of the rectangular upper vessel. The two upper drums were connected with the lower or mud-drum by two series of tubes divided by a fire-brick baffle-wall, the flame going up the

front set and down the back set of tubes to the chimney-flue. This boiler proved superior to the others, but the feed-water had either to be delivered into the mud-drum (thus constantly stirring up the mud), or else into one of the upper drums where the heat was so great that the deposit from the feed-water would be baked into a hard scale.

Fig. 2.

The remedy for this defect was found in the present form of the Stirling boiler, in which the water-tubes are expanded directly into the drums (Figs. 1 and 2). There are no flat surfaces, no stays, and no headers, with their numerous hand-holes.

The Stirling boiler is practically self-cleaning, because the water is fed into the back upper-drum, and descends with a slow motion of 6 inches per minute to the mud-drum through the back group of tubes, which have an area 100 times greater than that of the feed-pipe. On entering the mud-drum the feed-water has reached the boiling-point corresponding to the pressure under which the boiler is working. The scale-forming matter, together with other solid matter held in suspension prior to the feed-water entering the boiler, is deposited on the bottom of the mud-drum from which it is readily blown-off. This arrangement ensures the supply of practically pure water to the front and middle groups of tubes, where the steam is made.

The horizontal fire-tube has deposits of coal-dust and fine ashes on the inside and of sediment on the outside; the common water-tube has deposits of sediment inside and coal-dust outside; and the Stirling water-tube has comparatively clean inside and outside surfaces. The Stirling boilers are giving the highest possible results that are opened only once in six months, and even then very little cleaning is necessary.

With regard to utilization of heat: the heat of the fire-gases is thoroughly exhausted before they leave the water-tube boiler, because (1) the fire-gases are caused to pass successively over the whole length of the three groups of tubes; (2) the water and fire-gases are divided into small sections by the tubes; (3) the form of the current of fire-gases is changed nine times in passing from the furnace to the chimney; (4) the heating-surface is comparatively clean, both inside and outside; (5) the heating-surface is thin; and (6) the gases leave the boiler where the feed-water enters, thus reducing the temperature of the escaping-gases and heating the feed-water with the waste heat. This renders it possible to reduce the temperature of the escaping-gases below the temperature of the steam.

The circulation of the water in the boiler is steady and thorough. Each tube has a separate outlet to the drums of the full size of the tube. Steam is made in the front and middle groups of tubes, and these tubes are inclined at the best angle for allowing the steam to pass freely and quietly to the steam-space without carrying water with it. The geyser-like action which takes place when water is boiled in a vertical tube entirely disappears when the tube is sufficiently inclined and working under pressure. The circulation of water between the front and middle drums passes freely through the numerous tubes which connect them below the water-level.

The tubes are divided into three groups, each of which is expanded into a separate drum. This arrangement allows for unequal expansion and contraction in each group, and the bends in the tubes provide for any unequal expansion in the individual tubes.

By opening four manholes, one in each drum, access is gained to the inside of every part of the steel-plates and to the ends and the interior of every water-tube. Openings are provided in the brickwork for the purpose of getting at the outside of the plates and water-tubes.

The quiet and thorough circulation of water, the large area of water-level, and the large capacity for steam and water, ensure a steady supply of dry steam.

The advantages of using high-pressure steam are widely recognized, and its use in modern engines is increasing rapidly. The merits of the water-tube as compared with other boilers for carrying high pressures, are conceded by the best engineering authorities; and the Stirling water-tube boiler commends itself because (1) there are no riveted joints exposed to the heat; (2) there are no flat surfaces, and, consequently, no stays are required; (3) the outside surfaces of the tubes are the only parts with which flame comes into contact; (4) the ends of the tubes are in water;

(5) there is no heating-surface above the water-line; (6) all the parts are of wrought steel; (7) the tube-plates are made thicker to allow for drilling the tube-holes; (8) expansion and contraction have been thoroughly provided for; (9) the water is divided into small sections; (10) the circulation is steady and thorough; (11) it is practically self-cleaning, so that the boiler seldom requires to be opened; and (12) there are only four joints to break to get access to every part of the boiler.

The numerous divisions of the water and fire-gases and the thin tube heating-surface, combined with the free circulation, ensure rapid and efficient steam-raising.

If from carelessness there should be a heavy fire in the furnace and the water left low, the worst that could happen would be rupture of one or more of the water-tubes, and the pressure would be relieved without serious consequences. Should such an accident occur to the tubes, others can readily be substituted without injuring any good tubes. The tubes are very easily removed and replaced, because they do not require to pass their whole length through the tube-hole as in other boilers, but only about an inch through the hole and back again, and they are clear. The tubes connecting the drums are readily fixed with the usual expanding tool.

Owing to its construction, the water-tube boiler will admit of a wide grate-surface, giving to each section its full quota of heat, and enabling the fireman to work the fire properly, which cannot be the case where the grate-surface is obtained by a long narrow fire-box.

There is a large draught-area resulting from the wide grate-surface and large tube-chambers. The nearly vertical position of the tubes and the arrangement of the brick baffle-plates in the tube-chambers ensure a good draught.

The combustion is good because (1) the air-supply to the grate is heated; (2) the oxygen and carbon are thoroughly mixed and heated to a high temperature on the large furnace; and (3) the combustion is completed in the combustion-chamber before the fire-gases reach the tubes. Radiation through the brick-setting is prevented by the erection of double walls with an air-space between. The air is drawn in through this air-space to the ash-pit, and the heat that would otherwise be lost by radiation is carried back into the boiler and utilized for heating the air for combustion.

The advantage of a thin heating-surface over thick plates is apparent. The thin clean tube-surface in a water-tube boiler cannot be injured by the most intense fire, so long as the tubes are full of water.

The Stirling water-tube boiler occupies less space than any other boiler, and the fire-room can be made any width without regard to the taking out of the tubes. The small space required for boilers and fire-room effects a saving in cost of ground, foundations, and buildings.

The Stirling water-tube boiler is supported on wrought-iron columns, and is sustained independently of the brickwork. It is free to expand and contract without affecting the brickwork, which may be removed and replaced without disturbing the boiler or connexions. The furnaces and grates can be arranged for coal, wood, bagasse, oil, or any other fuel.

The following are the principal dimensions of Stirling boilers recently erected :—

Locality.	Kilmarnock.	St. Enoch's Station, Glasgow.	Motherwell.
Heating-surface, square feet	621	2,587	4,370
Grate-surface, do.	11	34	90
Ratio of heating to grate-surface	56	76	48
Number of water-tubes	78	264	391
Diameter of water-tubes, inches	3¼	3¼	3¼
Diameter of drums, feet	3	4	4
Length of drums, do.	5	9	11
Floor-space occupied, width, feet	7¼	11¼	14
Do. do. depth, do.	14	15½	17

Mr. JAMES S. DIXON (Glasgow) wrote that Babcock and Wilcox water-tube boilers had been in use at Hamilton Palace colliery for about a dozen years. For about ten years they raised large supplies of steam, and gave no trouble. During this time they were fed with water from the pit, which contained little or no free acid, and deposited almost no scale. Since then, water containing impurities of both kinds got access from the old workings of an adjoining colliery, and the tubes became both pitted and encrusted. In order to obviate this damage, water for the boilers is now being pumped from the river Clyde. The tubes in these boilers lie at an inclination, and the incrustation is chiefly found in the lower 4 or 5 feet of the tubes next to the mud-drum, and may be due to the lower temperature of the furnace-gases at that point, as the circulation through the water-tubes must be the same throughout. Mr. Stirling claimed that the back range of tubes of his boiler being placed in an upright position obviated this difficulty. If this be so, one difficulty with bad water—and a serious one—will have been overcome. Mr. Stirling also claimed that the water-tubes were kept clean, both inside and outside. There is no doubt that an upright tube will not carry dirt outside like a flat or sloping

one, and that cleanliness has a marked effect on the steam-raising powers of a water-tube boiler. He (Mr. Dixon) would like to have an explanation as to how the current of fire-gases is changed in direction so many as nine times from the furnace to the chimney. If the steam passes off so quietly as described, priming would be prevented—a most desirable result. The experience of some engineer who had used the Stirling boiler would be valuable on that point. If the case of removing a bad tube was in practice, as described, it was a favourable point, as in the Babcock and Wilcox boilers great trouble was often experienced in withdrawing a bulged tube through the header. There was no doubt that the water-tube form of boiler was a move in the right direction, especially in these days of high pressures; but his experience was that good water was a *sine quâ non*. Of course the matter of first cost was a consideration, and it would be desirable to know the cost of the Stirling boiler, say, per 100 indicated horse-power.

Mr. James Frew (Dunaskin) asked whether Mr. Stirling had experienced any difficulty with scale adhering to the water-tubes leading from the back top-drum to the mud-drum, and whether the providing of one hundred times the feed area in the water-tubes leading to the mud-drum entirely overcame the trouble of scale deposit, or only reduced it? In the case of the boiler being fired with coal or dross, when the temperature of the water-tubes was lowered during the cleaning of the fires, he thought it probable that leakage at the tube-ends would take place. He asked whether there was any trouble from leakage at that point? It seemed to him, on the whole, that the Stirling boiler was ingenious and well-arranged, and that it would secure the utilization of the heat given off from the fuel.

Mr. R. D. Munro (Glasgow) said that Mr. Stirling need not be surprised if the steam-users of this country were somewhat slow in taking up any type of water-tube boiler. He thought that engineers were justified in being cautious before giving up the Lancashire and other well-known types for a design so remote from preconceived ideas of what a boiler should be. It was just possible that they were all wrong in their estimate of Lancashire boilers, but they had before them the fact that they had done nearly all the steam-raising in this country for upwards of thirty years, and that hitherto it had not been proved that boilers of the water-tube type could surpass them in efficiency when working under the same conditions. He also felt certain that the water-tube boiler would never excel the Lancashire in the matter of durability. They had been accustomed to Lancashire boilers carrying considerable pressures for twenty-five years with the greatest safety, and they were naturally anxious to learn how the

new types would behave under a lengthened working-test before they took them into their confidence. What Mr. Stirling had said would tend to direct attention to this very important question. The steam-pressures employed were now much above the average of former years, and they were rapidly being raised to a point at which it was perhaps questionable if Lancashire boilers could be worked satisfactorily. When they considered the enormous weight and bulk of a Lancashire boiler, 30 feet in length by 8 feet in diameter, suitable for pressures from 150 to 200 lbs. per square inch, they found themselves face to face with difficulties in transit and greatly increased expenditure for foundations, and it was just probable that the wear and tear, which, with lighter boilers and lower pressures, was very gradual, might be such, under the altered circumstances, as would detract seriously from the usual lifetime of the Lancashire boiler. Whether the water-tube, so-called, or some other form of sectional boiler, would become the type of the future had yet to be determined; but it was evident that some of the leading boiler engineers were of opinion that a sectional boiler could be made which possessed all the advantages of the Lancashire boiler, and at the same time be free from the objections as to great weight, etc., already referred to. In his own experience of water-tube boilers, he had found that they had worked fairly satisfactorily, provided the feed-water was good and the steam requirements were steady, but in paper-mills, dye-works, and other factories where the demands for steam were of an intermittent nature, they had not proved so serviceable as boilers of the Lancashire type. This, he thought, was largely due to the small steam disengaging surface of the water-tube boiler in proportion to its heating-surface. Water-tube boilers were seriously affected by the forced firing that was too frequently resorted to when extra steam was required, the result being that foaming or priming of a violent nature very often occurred, causing great loss of economy, besides endangering the safety of engine-cylinders, piping, etc. Great improvements had been effected in the water-tube boiler within recent years, and as they were being adopted by their own and foreign governments for men of war, he thought this was likely to give an impetus to their use on land. The Stirling boiler was a distinct departure from those used in this country, and as it had already attained great success in the United States of America, it was just probable that they might find it had been designed to overcome the troubles incidental to the use of impure feed-water and priming, which, in certain circumstances, were the serious drawbacks of other water-tube boilers. In the Stirling boiler there was a combined steam-boiler and economizer, the first two banks of tubes forming the

boiler proper, and the last bank of tubes forming an economizer, in which the feed-water was heated by gases of low temperature. By this disposition of the tubes, he understood Mr. Stirling to say that he could more effectually utilize the heat of the furnace-gases, there being no difficulty in doing so to a degree even below the temperature of the steam generated. The same claims, however, had been made by the various manufacturers of the economizers used in connexion with Lancashire boilers, but there was nothing to be gained by reducing the gases to such a low temperature; and, except in cases where there was a very large chimney or some system of artificial draught, it was impossible to work satisfactorily with gases reduced below that of the steam generated. The minimum chimney temperature in good practice was about 400 degs., but the average, unfortunately, was as high as 600 degs. He had heard of some installations for forced draught, in which the furnace-gases were so thoroughly utilized that they escaped into the chimney at a temperature of about 200 degs.; but, speaking generally, it had been found that, except in special circumstances, there was quite as much economy in maintaining good draught by an expenditure of heat at the base of the chimney, as in providing steam for fans, or jets for artificial draught. The arrangement for feeding the Stirling boiler was an ingenious one, and whilst he would like to see more of it before deciding as to its merits, he was inclined to consider it as being favourable to prevention of trouble in the use of bad feed-water, and thereby likely to promote economy and durability. He understood Mr. Stirling to say that water-tube boilers were now the rule in America, as well as on the Continent, and that Britain was alone in her advocacy and continued use of fire-tube boilers. There must, he thought, be some mistake in this, as he was regularly in receipt of reports from an American company, which inspected some 40,000 steam boilers annually, and his impression at the moment was that about nine-tenths of these were of the ordinary fire-tube type. His experience of Continental countries also went to show that fire-tube boilers still held their own, notwithstanding the numerous patents that had been taken out for water-tube boilers, both in France and Germany. With regard to the large number of explosions in America, compared with British experience, he might state that, according to published statistics, there had been upwards of 300 explosions in the United States of America during the year 1894, many of them being of a most disastrous nature, whilst the average number of actual explosions in this country did not exceed 40 per annum. He thought that these facts pointed to the necessity for improved supervision on the part of

American engineers. The boilers, it might be thought, were badly designed or badly constructed, but he was of opinion that the explosions were more the result of the reckless manner in which the boilers were worked and attended to, than to defects in their design and construction. The water-tube boiler was generally spoken of in this country as an American invention, but it might be of interest to Mr. Stirling to know that the late Mr. Rowan, the father of a prominent member of this Institute, if not the first to introduce the water-tube boiler, was at least one of the pioneers. He designed the boilers of the steamship "Propontis," and although these boilers were not successful, they formed a basis or starting-point from which succeeding water-tube boilers were constructed. In conclusion, he would like to know Mr. Stirling's opinion as to the method of discharge adopted in the Thorneycroft boiler. The tubes were arranged so as to deliver through the steam space, and it was said that this added greatly to the efficiency of the boiler. He would also be indebted to Mr. Stirling if he would give some statistics as to the performance of the Stirling boiler. The members were all fairly conversant with the duty and efficiency of the Lancashire and other fire-tube boilers, and it would therefore be very interesting to have full particulars of evaporative tests of the Stirling boiler, so as to enable them to judge of its duty and efficiency. He ventured to express the opinion that good though this boiler might prove to be, it would never surpass the Lancashire boiler in efficiency, but, inasmuch as it was better adapted for carrying very high steam-pressures, and possessed advantages over some other designs of water-tube boilers, it would probably be taken up to some considerable extent by the more enterprising steam users.

Mr. HENRY AITKEN (Falkirk) agreed with Mr. Stirling that the water-tube boiler was the boiler of the future, but there were circumstances where it would not be so, *e.g.*, where the water was bad. He had had great experience with bad water and scaled boilers and tubes. In connexion with one where Field tubes were used, he had obtained good results, but in the course of a fortnight the whole of the tubes had to be taken out. In doing so he cleared the boiler with iron plugs, which extended 6 inches into the gas space below, and 6 inches into the water in the boiler, and curiously enough many of your results were got as if the Field tubes had been in position. He [illegible] in the Stirling boiler the difficulty of bad water was got rid of. But for all general purposes where the water was good he had no doubt, particularly for high pressures, that the water-tube boiler was the boiler of the future.

Mr. STIRLING said that the current of fire-gases left the furnace in a solid mass, and (1) was divided in strips in passing in among the tubes; (2) it followed the front group of tubes in the form of a rectangle minus the circles of the tubes; (3) it was divided into strips in passing out of the first group of tubes; (4) it passed across the space between the first and second group of tubes in a solid mass; (5) it was divided into strips in passing in among the second group of tubes; (6) it followed the middle group of tubes in the form of a rectangle minus the circles of the tubes; (7) it was divided into strips in passing from the second to the third group of tubes; (8) it followed the third group of tubes in the form of a rectangle minus the circles of the tubes; and (9) it was divided into strips in passing out from the third group of tubes to the chimney. The Stirling cost less than any other boiler, and could be supplied for less than 20s. per indicated horse-power. There was no difficulty with scale or leaky tubes in the Stirling boiler. He (Mr. Stirling) had found no fault with the cautious feature of the Scotch character, but in these days of keen international competition engineers found it necessary to be very wide-awake in exercising their judgment in adopting the best methods and appliances. The Stirling surpassed the Lancashire boiler when working under the same conditions, and it was much more durable than the Lancashire for modern high-pressures. Although the Lancashire boiler had done nearly all the steam-raising in this country for upwards of thirty years at low-pressures, that was no evidence that it would continue to prove satisfactory at the modern high-pressures. Mr. Munro questioned whether Lancashire boilers could be worked satisfactorily at high-pressures. The Stirling boiler overcame all the difficulties which Mr. Munro had found in Lancashire and in other water-tube boilers: (1) It would work satisfactorily with bad feed-water; (2) the numerous divisions of the water and fire-gases, and the comparatively clean, thin, heating-surface, combined with the large steam-disengaging surface, the free circulation, etc., ensured rapid and efficient steam-raising, and enabled the Stirling to respond much more promptly to intermittent demands for steam than the Lancashire boiler or other types of water-tube boilers; and (3) the Stirling boiler did not foam or prime. The ordinary way of producing draught by means of heated air in a chimney was wasteful. Years ago, mines were ventilated in this way, but the fan had been universally adopted instead. Fans were used with great economy in connexion with steam-boilers in many cases, particularly on shipboard. The most economical way to operate a boiler-plant was to reduce the temperature of

the chimney-gases as much as possible, and to blow the fire with a fan driven by an engine. For large power-plants, the water-tube system was being almost universally adopted in America. With reference to explosions, although there are over twelve hundred Stirling boilers at work in America there had never been an explosion, nor had a single person been hurt by them. He (Mr. Stirling) was familiar with the Rowan boiler that failed on the "Propontis," and also with the boiler that failed on the "Montana." He considered that it was very objectionable to have any heating-surface above the water-line. He had not devoted attention to the making of tests, but others had been kind enough to give him their experience. Mr. Eckley B. Coxe, President of the American Society of Mechanical Engineers, on April 6th, 1894, wrote:—"We have been running the Stirling boilers at Oneida continuously since May last, and can give no information as to their weak points, as we have as yet found none. We have done nothing to them. We have made many experiments with them, in one case running one of them (150 horse-power) up to 240 horse-power without any bad results to the boiler. They have not leaked or sprung, and seem as good as when put in." Mr. Philip D. Armour, of the Armour Canning Co., Chicago, wrote, on April 3rd, 1894:—"We have purchased from the Stirling Company over £23,500 worth of boilers within the past eighteen months, and they have given entire satisfaction. We find them a great saving in coal."

The discussion was then adjourned.

Mr. W. C. MOUNTAIN read the following paper on "The Design, Efficiency, and Application of Electric-motors for Transmission of Power in Mines":—

THE DESIGN, EFFICIENCY, AND APPLICATION OF ELECTRIC-MOTORS FOR TRANSMISSION OF POWER IN MINES.

By W. C. MOUNTAIN.

Introduction.

The writer trusts that he will be excused for commencing this subject from the beginning, and he proposes in the first instance to give an explanation of the various units used by electrical engineers, and their practical application to an electrical transmission of power-plant.

The practical units corresponding to foot-pounds, pressure per square inch, quantity of steam, and loss in steam-pipes, are the volt, the ampère, and the ohm.

The volt is the unit of electro-motive force. It is usually written E.M.F. and is equivalent to the difference of potential between two points. The legal volt is the electro-motive force which maintains a current of 1 ampère in a conductor whose resistance is the legal ohm. The volt is equal to 100,000,000 C.G.S. units of electro-motive force or about 7 per cent. less than the electro-motive force of a standard Daniell cell. The Daniell cell is a primary battery consisting of copper and zinc elements, the copper being immersed in a solution of sulphate of copper, and the zinc in a solution of sulphuric acid and water, the two elements being separated by a porous pot. The electro-motive force of a standard Daniell cell is about 1·1 volts.

The ampère, the unit of current, is described practically as that amount of current which decomposes 0·00009324 gramme of water per second, or deposits 4·025 grammes of silver per hour; 386·4 ampères will deposit 1 pound of copper per hour.

The ohm, the unit of resistance, is equal to the resistance of a column of pure mercury 1 square millimetre in section and 106 centimetres long at the temperature of melting ice, and is approximately equal to the resistance of 129 yards of copper wire $\frac{1}{16}$ inch in diameter.

In constructing a dynamo or considering the type required for any specific purpose it is usual to design the machine to give a certain output in watts or to require a certain number of watts for any particular work.

Watts are represented by multiplying the ampères and volts together, the watt being a volt-ampère. For instance, assuming that a dynamo gives a current of 200 ampères at an electro-motive force of 100 volts, these multiplied together equal 20,000 volt-ampères or watts. As 746 volt-ampères or watts are equal to 1 electrical horse-power, and further correspond with the 33,000 foot-pounds per minute used in engineering calculation as the standard horse-power, it is necessary in order to determine the horse-power required to drive a dynamo of any given capacity, to divide the total watts by 746, which gives the electrical horse-power required to drive the machine. All dynamos naturally require more power to drive them than the actual output at the terminals of the machine represented in watts, because a certain amount of current is required to excite the machine, and a certain loss occurs due to the resistance of the armature, and a further loss due to the friction of the shaft and bearings.

In large dynamos, the commercial efficiency, *i.e.*, the output in watts at the terminals compared with the power required to drive, is as high as 96 per cent., but in smaller machines the efficiency is lower.

For the particular machine mentioned above, giving an output of 20,000 watts, the efficiency may be taken at 90 per cent. The formula for calculating the horse-power required to drive a dynamo giving an output of 20,000 watts (commonly called a 20 units machine: 1,000 watts being known as a Board of Trade unit) is as follows:—

$$\frac{20{,}000 \times 100}{746 \times 90} = 29{\cdot}78 \text{ horse-power required to drive the dynamo.}$$

This power is usually described as the effective or brake horse-power. The effective or brake horse-power of a steam-engine varies with the class of engine and with the size of the engine, but in an engine such as would be required for driving a dynamo of the above size, viz., 20 units, it would not be safe to estimate the efficiency of the engine at more than 75 per cent., consequently the formula becomes

$$\frac{29{\cdot}78 \times 100}{75} = 37{\cdot}7 \text{ indicated horse-power required to drive the dynamo.}$$

A general description of the construction of a dynamo may be of interest, and the writer therefore proposes to explain the construction of an ordinary Gramme type dynamo of the two-pole pattern. Fig. 1, Plate I., is a section through the armature, commutator, and bearings of the machine. Fig. 2, Plate I., is an end view, showing the brush-holders and general connexions to the field-magnets and outer circuit.

A dynamo consists of three essential parts—(1) the armature which is the revolving portion of the machine; (2) the field-magnets which are the stationary portions of the machine; and (3) the commutator from which the current is collected by the brushes.

The core of the armature *A* is composed of a large number of charcoal-iron discs insulated from each other either by vulcanite or paper. The discs are firmly cramped together and slotted, and keyways are usually cut in them, into which a gun-metal driver is fitted, long bolts cramping the whole armature together. The armature is then turned on the outside, and the armature-conductors are wound upon it, these conductors being coils forming a continuous ring around the core, branches being taken off at various points and joined to the bars of the commutator.

The field-magnets *F.M.* in the particular machine illustrated are of the horse-shoe type. The section of the magnet is proportioned to the section of the armature, so as to obtain the best results, and it has been found in practice that it is necessary to make the section of the magnets very heavy in order to obtain a high induction in the armature, to prevent sparking and to enable the machine to work without heating.

The field-magnet coils *F.M.C.* should be wound upon independent bobbins, so that they can be removed from the machine in case they require repairs. It is also desirable to have the magnets placed above the armature, so as to prevent oil from falling upon them.

The commutator *C* is made of hard-drawn copper sections; the various sections are insulated with mica, and then mounted upon a gun metal or cast-iron sleeve, and held in position by the V-collar at each end. It is very necessary to pay particular attention to the commutator in attending to a dynamo. In a good machine the commutator should run with little or no sparking, unless, of course, the machine is wound for a very high tension, in which case it is very difficult to prevent a certain amount of sparking.

An electro-motor is constructed on the same lines exactly as a dynamo, and the explanation respecting the construction of a dynamo applies equally to the construction of a motor.

We have now to consider the class of dynamo and motor most suited for transmission of power, and to explain the various methods of winding the field-magnets. Three methods of winding are in general use, viz., series, shunt, and series-and-shunt or compound winding.

A series-wound dynamo (Fig. 3, Plate I.) is seldom used for transmission of power, because it does not excite very readily. Further, the

machine does not give a sufficiently constant electro-motive force, as the excitation of the magnets depends entirely upon the amount of current passing through the outer circuit, and when little current is passing it follows that the magnetism must be less, and the electro-motive force of the machine is also less.

Shunt-wound dynamos (Fig. 4, Plate I.) are largely used for transmission of power; but with this type of machine the electro-motive force drops as the work is required from it, because the excitation of the field magnets, *i.e.*, the current flowing through the shunt-coils, which are connected between the positive and negative terminals, is practically constant, consequently the electro-motive force drops, owing to the resistance of the armature as the load is increased.

A compound, or series-and-shunt-wound dynamo (Fig. 5, Plate I.), is suitable for transmission-of-power installations, particularly where a large number of motors have to be driven from one machine. This system is really a combination of a series and a shunt-wound dynamo, the shunt-coils maintaining a practically constant degree of magnetization in the field-magnets, and this is increased, owing to the current flowing through the series wire, and passing to the outer circuit, which slightly increases the strength of the magnets, and overcomes the loss in volts due to the resistance of the armature, or in other words, the magnets of the machine give a slightly higher electro-motive force in the armature, but not in the outer circuit, as the higher volts are absorbed by the resistance of the armature, and a constant electro-motive force under varying loads is consequently maintained.

The series-motor is largely used for transmission of power where the machines have the constant attention of a workman, or where the load is not likely to vary. This type of machine has considerable starting power, it does not run at a constant speed under varying loads, but where the load is constant this defect is immaterial.

A shunt-motor has the important property of running at a constant speed under a varying load; it is therefore largely used where the load may vary, and the motors are not being constantly watched.

The compound-motor is very seldom made, but it has the important properties of both the series and shunt-motors, viz., it can start under considerable loads, and, owing to the shunt-winding, it will maintain a practically constant speed.

From these explanations, it will be at once seen that the series-motor is particularly suitable for haulage or pumping machinery. In mines, where a number of small pumps, driven by electric-motors, are run with-

out constant attention, and the pumps are liable to lose their water, it is necessary to adopt the shunt-motor.

Electrical transmission of power is particularly suitable for collieries or for general transmission of power work for the following reasons:—(1) motors occupy a very small amount of space compared with steam-engines; (2) they require very little attention in working; (3) they are extremely efficient, and with light loads the efficiency is very much greater than in any other type of motor, such as a steam-engine; (4) no loss of power occurs in the cables unless the motors are working, and the loss becomes less as the load is decreased; and (5) they can be placed at long distances from the point where the current is generated, and the cables can be laid to the motors much more easily than steam, water, or compressed-air pipes.

At this point it may be desirable to explain the formulæ which are used to ascertain the loss of power in an electric cable. The two losses to be ascertained are the loss in volt-ampères or watts and the loss in volts only. The formula used for finding the loss in watts or volt-ampères is:—$C^2 \times R$, in which C is the current in ampères and R the resistance of the cable.

The following example will explain the use of the formula:—To find the loss in watts in a 7/15 cable, *i.e.*, a cable composed of 7 wires of No. 15 legal standard gauge, one mile in length, with a current of 29 ampères (*i.e.*, approximately 1,000 ampères per square inch of section of copper) the resistance of one mile of 7/15 cable, being 1·498 ohms.

$$C^2 R = 29^2 \times 1{\cdot}498 = 1{,}259{\cdot}811 \text{ watts.}$$

To ascertain the loss in volts E, the resistance R of the cable must be multiplied by the current C in ampères flowing through it.

To find the loss in volts E in a cable composed of 7 wires of No. 15 legal standard gauge, one mile in length, with a current of 29 ampères (*i.e.*, approximately 1,000 ampères per square inch of section of copper), the resistance of one mile of 7/15 cable being 1·498 ohms:

$$C \times 1{\cdot}498 = 29 \times 1{\cdot}498 = 43{\cdot}442 \text{ volts.}$$

A practical rule, and one which will be useful to engineers, is to remember that the loss in the cable is approximately $2\frac{1}{2}$ volts per 300 feet of cable where the current-density is 1,000 ampères per square inch of section of copper.

Table I. gives the sizes of all standard-wire conductors, together with the diameter of each wire, sectional area in square inches and square millimetres, weight per mile in lbs., resistance in ohms, and carrying capacity in ampères.

Too great a stress cannot be laid upon the importance of having good cables for use in mines, thoroughly well insulated, and armoured if necessary as a mechanical protection. The various substances used for the insulation of cables are :—Pure indiarubber, vulcanized indiarubber, vulcanized bitumen, bitumenized fibre and paper.

Pure indiarubber can only be used in very dry positions and cannot be recommended for use in mines under any conditions: it is impossible to get anything like proper insulation, as the rubber is laid on in a band or ribbon, and the joint cannot be made water-tight.

Vulcanized indiarubber is an extremely satisfactory insulating-material for low-tension currents, *i.e.*, where the pressure does not exceed 500 volts, cables with an insulation resistance of 600 megohms per mile are very suitable.

Vulcanized bitumen is a satisfactory insulating-compound. The bitumen after being laid on the cables is vulcanized, braided, and usually drawn into a leaden pipe.

Bitumenized fibre is largely used for cables which have to be very roughly handled. The cable is insulated with a number of braidings of fibre which are steeped in bitumen. This cable must be drawn into a leaden pipe, as the fibre will absorb damp unless so protected. The lead-covered cable can, if necessary, be placed in damp positions. It is always desirable to ascertain, before putting a cable of this description into a mine, whether the water is likely to act upon the lead and destroy it. There have been many instances in which lead-covered cables have been very quickly destroyed by the action of water.

The paper insulation is very similar to the bitumenized fibre, and should also be drawn into leaden pipes for protection.

For use in shafts or on underground engine-planes it is necessary in almost all cases that the cables should be armoured, as a mechanical protection, either with galvanized iron wires or with steel tapes. This protection does not add very materially to the cost, and it adds to the strength of the cables, which can then support their own weight without danger.

It may be generally taken that, for installations in which the electro-motive force exceeds 500 volts, the insulation-resistance of the cables should not be less than 1,000 megohms and preferably 2,000 megohms per mile. The insulation-resistance is the resistance due to the insulating-compound, that is, the resistance between the conductor and the earth. This resistance is usually measured by immersing the cable in salt-water for twenty-four hours, and then measuring the resistance between the conductor and the earth.

TABLE I.—PARTICULARS OF CONDUCTORS.

Legal Standard Gauge.	Diameter of each Wire.		Diameter of the Strand.		Sectional Area.		Weight per Mile.	Resistance.	Carrying Capacity.
	Inches.	Millimetres.	Inches.	Millimetres.	Square Inches.	Square Millimetres.	Pounds.	Ohms.	Ampères.
7/22	·028	·711	·084	2·13	·0044	2·838	90	9·908	4·4
7/21	·032	·813	·096	2·44	·0057	3·703	117	7·596	5·7
7/20	·036	·914	·108	2·74	·0072	4·690	148	5·998	7·2
7/19	·040	1·02	·120	3·04	·0089	5·786	183	4·861	8·9
7/18	·048	1·22	·144	3·66	·0129	8·321	263	3·380	12·9
7/17	·056	1·42	·168	4·27	·0176	11·35	358	2·477	17·6
7/16	·064	1·62	·192	4·88	·0230	14·83	468	1·895	23·0
7/15	·072	1·83	·216	5·49	·0291	18·77	592	1·498	29·1
7/14	·080	2·03	·240	6·10	·0359	23·16	731	1·214	35·9
7/13	·092	2·34	·276	7·01	·0475	30·64	967	·9179	47·5
7/12	·104	2·64	·312	7·93	·0607	39·16	1235	·7183	60·7
7/11	·116	2·94	·348	8·84	·0755	48·70	1537	·5776	75·5
7/10	·128	3·25	·384	9·75	·0919	59·28	1870	·4744	91·9
19/22	·028	·711	·140	3·56	·0120	7·741	244	3·634	12·0
19/21	·032	·813	·160	4·07	·0156	10·06	318	2·795	15·6
19/20	·036	·914	·180	4·57	·0198	12·77	403	2·202	19·8
19/19	·040	1·02	·200	5·08	·0244	15·74	497	1·786	24·4
19/18	·048	1·22	·240	6·10	·0351	22·64	714	1·243	35·1
19/17	·056	1·42	·280	7·11	·0479	30·90	975	·9103	47·9
19/16	·064	1·62	·320	8·13	·0625	40·32	1272	·6976	62·5
19/15	·072	1·83	·360	9·14	·0791	51·03	1610	·5512	79·1
19/14	·080	2·03	·400	10·16	·0976	62·96	1987	·4467	97·6
19/13	·092	2·34	·460	11·68	·129	83·22	2625	·3380	129
19/12	·104	2·64	·520	13·22	·165	106·4	3358	·2642	165
19/11	·116	2·94	·580	14·73	·205	132·2	4173	·2126	205
19/10	·128	3·25	·640	16·26	·250	161·2	5090	·1744	250
37/19	·040	1·02	·280	7·11	·0477	30·77	971	·9140	47·7
37/18	·048	1·22	·336	8·53	·0686	44·25	1396	·6355	68·6
37/17	·056	1·42	·392	9·96	·0934	60·25	1901	·4667	93·4
37/16	·064	1·62	·448	11·38	·122	80·70	2483	·3574	122
37/15	·072	1·83	·504	12·80	·154	99·35	3134	·2831	154
37/14	·080	2·03	·560	14·22	·191	123·2	3887	·2283	191
37/13	·092	2·34	·644	16·36	·252	162·5	5129	·1730	252
37/12	·104	2·64	·728	18·49	·322	208	6554	·1354	322
37/11	·116	2·94	·812	20·63	·401	259	8160	·1087	401
37/10	·128	3·25	·896	22·76	·488	315	9931	·0893	488

TABLE I.—*Continued.*

Legal Standard Gauge.	Diameter of each Wire.		Diameter of the Strand.		Sectional Area.		Weight per Mile.	Resistance.	Carrying Capacity.
	Inches.	Millimetres.	Inches.	Millimetres.	Square Inches.	Square Millimetres.	Pounds.	Ohms.	Ampères.
61/18	·048	1·22	·432	10·97	·113	73·00	2300	·3859	113
61/17	·056	1·42	·504	12·80	·154	99·35	3134	·2831	154
61/16	·064	1·62	·576	14·63	·201	129·6	4091	·2169	201
61/15	·072	1·83	·648	16·46	·255	164·5	5189	·1710	255
61/14	·080	2·03	·720	18·29	·315	203	6410	·1384	315
61/13	·092	2·34	·828	21·03	·416	268	8467	·1048	416
61/12	·104	2·64	·936	23·77	·532	343	10820	·0819	532
61/11	·116	2·94	1·044	26·52	·661	426	13450	·0659	661
61/10	·128	3·25	1·152	29·26	·805	519	16390	·0541	805

EFFICIENCY OF MOTORS.

One of the great advantages of adopting electric transmission of power is the fact that a dynamo or a motor is self-regulating, *i.e.*, the dynamo only requires power to drive it in accordance with the work which it is called upon to perform, and a motor only requires current to drive it in accordance with the work which it is doing. Of course in each case the mechanical loss, due to the friction of the shaft in the bearings, is constant, but as this loss is so slight in an efficient and well-constructed machine, it forms a very small percentage of the total power.

It is of great importance that the dynamo and the motor should be of sufficient size for the work which they are called upon to perform. When a mining engineer is ordering an engine for a haulage or winding-plant or for pumping purposes, he usually orders one which is capable of doing at least twice the work normally required, but if he is ordering a motor or dynamo there is a very strong inclination to order a machine just capable of doing the work. There is no doubt that the breakdowns of electric plant which have occurred in the past have been due, not so much to defects in the machinery, but to the machinery being far too small for the work. Electrical engineers are to a great extent to blame for such failures, as they have been extremely deficient in knowledge of the actual requirements of mines, and further, owing to severe competition, they have no doubt supplied machines barely capable of performing the work required.

Table II. gives the watts required per horse-power at the main terminals for motors of various sizes. Assuming that it is required to

drive a motor at a distance of say 1,320 feet from the dynamo, and that the motor is to give 25 effective horse-power with an efficiency of 85 per cent., then 25 effective horse-power × 878 watts per horse-power = 21,950 watts required at the terminals of the motor. The voltage at which it is proposed to work the motor must then be assumed, and in this case it will be taken at 300 volts. The total watts, 21,950 ÷ 300 = 73·16 ampères, required to feed the motor.

TABLE II.

Effective Horse-power of Motor.	Commercial Efficiency, per cent.	Watts per Horse-power required at terminals.
3	70	1,066
5	75	995
15	80	904
25	85	878
40	88	848
50 and upwards ...	90	825

On referring to Table I. it will be seen that the required cable (assuming a density of 1,000 ampères per square inch) is one composed of 7 wires of No. 11 legal standard gauge, the sectional area being 0·0755 square inch. This cable will carry 75·5 ampères (at 1,000 ampères density), and has a resistance of 0·5776 ohm per mile; the length of the cable is ½ mile, *i.e.*, to the motor and back, and the resistance will be 0·288 ohm. The loss of watts in this cable will be $(73{\cdot}16)^2 \times 0{\cdot}288$ ohm resistance = 1,546 watts. If the watts lost in the cable be added to the watts consumed, the sum will give the watts required at the terminals of the dynamo, viz.:—

Watts consumed at motor ...	21,950
„ loss	1,546
Total	23,496

The generating dynamo must therefore be capable of giving an output of 23,496 watts. If the total number of watts, including the loss in the cable, be divided by the current, viz., 73·16 ampères, this will give the voltage required at the dynamo-terminals, thus

$$\frac{23{,}496}{73{\cdot}16} = 322 \text{ volts required at the dynamo-terminals.}$$

The effective horse-power given off at the motor-spindle can be found by reference to the formula already given, viz.:—

$$\frac{23{,}496 \times 100}{746 \times 85} = 37 \text{ effective horse-power required to drive the dynamo.}$$

This power represents an efficiency of 67·6 per cent.

We have now, however, to consider the loss in the engine, *i.e.*, between the indicated and the brake horse-power, and assuming the engine to be of a high-class, the loss should not be more than 15 per cent., or say 85 per cent. efficiency. The power required to drive the dynamo will then be: $(37 \times 100) \div 85 = 43{\cdot}5$ indicated horse-power.

The total efficiency between the indicated horse-power and the effective horse-power on the motor-spindle is 57·4 per cent., by the following formula: $(25 \times 100) \div 43{\cdot}5 = 57{\cdot}4$ per cent.

An important point for consideration is the best system of reducing the speed of the motors down to the speed required on the haulage, winding, pumping, or other machinery. The various methods of reducing speeds are as follows:—

(*a*) *Spur-gearing.* If spur-gearing be adopted, it should be exceedingly well made, the pinion and wheel, on first-motion shaft should be machined, or the pinion may be of steel accurately machined, and the wheel may be of cast-iron fitted with wooden teeth (a mortice wheel), otherwise the noise is very objectionable, and the excessive vibration which is created, even with the best gearing, is very liable to cause damage to the motor-armature.

(*b*) *By bevel-gearing*, where motion is required at right angles. This class of gearing is as objectionable as spur-gearing unless carefully machined.

(*c*) *Worm-gearing.* If worm-gearing is adopted, the worm should be made of case-hardened wrought-iron or steel, and the worm-wheel should be made of phosphor-bronze, very accurately machined. It is always advisable to use double-threaded and, if possible, treble-threaded gearing, otherwise the friction is considerable, but if good means of lubrication are provided and the power employed is not too great, this class of reduction-gear has been found to give most satisfactory results; an efficiency as high as 85 per cent. and even higher has been on many occasions obtained, a result equal to that given by the best spur-gearing.

(*d*) *A combination of either ropes or belts and spur-gearing.* This system is probably the best type of reduction-gearing that can be used: the ropes or belts act as an elastic medium and take all vibration off the motor-shaft, and the further reduction in speed is obtained by the spur-gearing.

(*e*) Another method is by belts or ropes alone, but this system is somewhat unsatisfactory and not generally adopted.

Objection has been taken to the use of belts or ropes for driving in mines. The writer does not consider that this objection is well founded, as he has known many instances where belts have run for three or four years underground, and are to all intents and purposes as good as on the day that they were put in.

Mining engineers are probably not generally acquainted with the loss that occurs in steam-pipes in mines. Mr. Hy. Davey carried out tests on a main steam-pipe 1,100 feet long, 7½ inches in internal diameter, and 1 inch in thickness. The main was used for conveying steam at a pressure of 45 lbs. per square inch to an underground pumping-engine at the Morton pit of the Clay Cross collieries. The pipes were covered with non-conducting composition. As a result of careful experiments, it was found that the condensation of steam in the main amounted to 500 lbs. of water per hour when the engine was standing, and 750 lbs. of water per hour when the engine was working, equivalent to 0·183 and 0·274 lbs. per square foot of surface of pipe per hour respectively.

At the Staveley Hill collieries, Mr. W. Worby Beaumont carried out experiments, which showed that, in a total length of 2,300 feet of pipe, 1,900 feet of which was underground, and covered in some instances to a depth of 2 feet with rough stuff left after cutting away the coal, the condensation in a pipe 6½ inches in diameter and 1⅛ inches thick, conveying steam at a pressure of 34 lbs. per square inch, was 600 lbs. of water per hour with the engine standing, and 400 lbs. of water per hour with the engine working. This condensation is equivalent to a condensation of 0·113 and 0·265 lbs. of water per square foot of surface of pipe per hour respectively. For uncovered pipes, of which unfortunately there are very many in mines, it may be very safely assumed that the condensation is not much less than 1 lb. of water per superficial foot of pipe exposed. If the pipe at the Staveley Hill colliery be taken as a fair example of underground work, a simple calculation shows that, assuming we evaporate 6 lbs. of water per lb. of coal, it is necessary to burn 233 lbs. of coal per hour in order to make up for the loss due to condensation; for six days per week, night and day, it will require slightly over 15 tons of coal per week, or, say, for 50 weeks, 750 tons of coal per annum, and this even at 5s. per ton means a loss of £187 10s. per annum.

The meeting was then closed.

To illustrate Mr W. C. Mountain's Paper on "The Design, Efficiency and Application of Electric-Motors," &c.

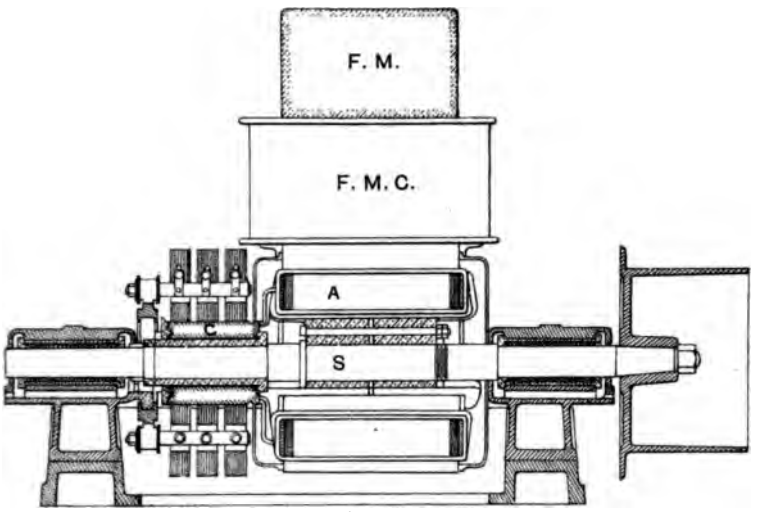


FIG. 1.

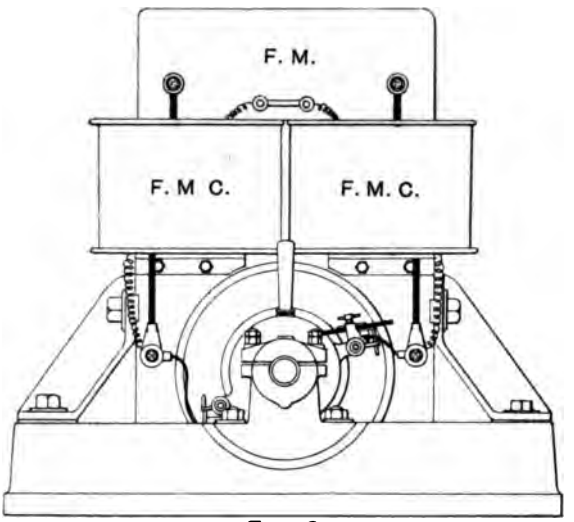


FIG. 2.

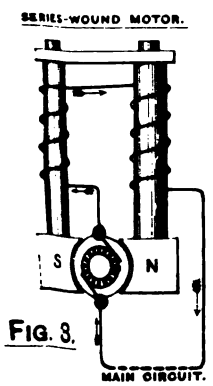


FIG. 3.

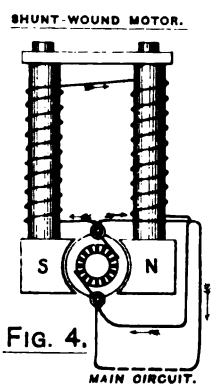
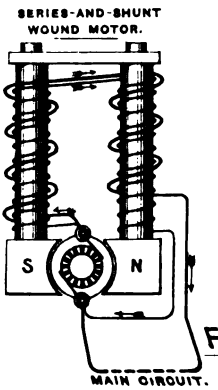


FIG. 4.



FIG

THE NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE, FEBRUARY 9TH, 1895.

MR. THOMAS DOUGLAS, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting, and reported the proceedings of the Council of that day.

The following gentlemen were elected, having been previously nominated :—

MEMBERS—

Mr. ALEXANDER AGASSIZ, Mining Engineer, Museum of Comparative Zoology, Cambridge, Massachusetts, United States of America.

Mr. THOMAS BATES, Colliery Manager, The Grange, Prudhoe-upon-Tyne.

Mr. ROBERT GILMAN BROWN, Mining Engineer, P.O. Box 946, Butte, Montana, United States of America.

Mr. WILLIAM CLIFFORD, Mining Engineer, Pittsburgh, Pennsylvania, United States of America.

Mr. JOHN S. CRAWFORD, Consulting Engineer, Kingston, Sierra County, New Mexico, United States of America.

Mr. JOHN HENRY DARBY, Steel Manufacturer and Engineer, Pen-y-Garth, Brymbo, Denbighshire.

Mr. DAVID EVAN DAVIES, Colliery Manager, Cwmaman Colliery, Aberdare.

Mr. THOMAS INGLEBY DYSON, Metallurgist, Mount Morgan, Queensland.

Mr. EDWARD EDWARDS, Colliery Manager, Maindy Pit, Ocean Coal Company, Ton Pentre, South Wales.

Mr. G. GRAVES GIFFORD, Mining Engineer, Perth, Western Australia.

Mr. GEORGE ERNEST GREGSON, Land Agent, Land and Mine Surveyor, 11, Chapel Street, Preston.

Mr. JOSEPH HADFIELD, Colliery Manager, Hulton Collieries, Chequerbent, Bolton, Lancashire.

Mr. DAVID HANNAH, Mining Engineer, Brynderwen, Ferndale, South Wales.

Mr. C. D. SULLIVAN, Mining Engineer, Bonang, Delegate, New South Wales.

Mr. JESSE WALLWORK, Assistant Mining Engineer, Haydock Colliery, near St. Helens, Lancashire.

ASSOCIATE MEMBER—

Mr. GEORGE BENEKÉ, Managing Director of an Explosives Manufacturing Company, 70, Gracechurch Street, London, E.C.

ASSOCIATE—

Mr. JOHN RIVERS, Master Shifter, Front Street, Tudhoe Colliery, Co. Durham.

STUDENT—

Mr. HENRY SCOTT STRATTON, Mining Student, Cramlington Collieries, Northumberland.

DISCUSSION UPON PROF. STROUD'S PAPER ON "MAGNETIC DECLINATION AND ITS VARIATIONS"* AND MR. JAMES HENDERSON'S PAPER ON "MAGNETIC DECLINATION IN MINES."†

Mr. ARTHUR BEANLANDS (Durham) said that the miner's compass had been generally used as an instrument for underground surveying for very many years, and possessed considerable advantages for that purpose. It was not, however, an infallible instrument, particularly in mines where much iron was used; and it was not attended with that degree of precision which would be required in special surveys, such as the defining of boundaries, etc. In the case of surveys requiring special care he advocated the use of the theodolite, in conjunction with the transit-instrument, to connect the surface with the underground survey. After the main lines of the survey had been laid down with the theodolite in this manner, the miner's compass was very useful in filling in details. In cases where the shaft was either very narrow, or so tortuous that the transit-instrument could not be used, he had sometimes depended on the position of two shafts (if the shafts were at a considerable distance from each other). He endorsed the desirability of paying attention to the annual variation in the magnetic declination, as in the course of a few years the difference became somewhat considerable. As the miner's compass could not be read to less than one-quarter or one-eighth of a degree, the diurnal variation scarcely made any perceptible difference.

Mr. J. A. RAMSAY (Littletown) generally endorsed the remarks made by Mr. Beanlands. In underground-surveying, the first great object to be attained was accuracy of departure from the bottom of the pit, and this certainly demanded the most special care and manipulation of the instrument used. To make an accurate survey of underground workings such an instrument as the theodolite was absolutely necessary, but he questioned whether the degree of accuracy which the workings of an extensive colliery demanded could be obtained from a base-line, the length of which could not by any possible means exceed a very small

* *Trans. Fed. Inst.*, vol. vii., page 268. † *Ibid.*, vol. viii., page 273.

fraction more than the diameter of the pit shaft. On the surface of the ground the base-line might be prolonged to practically any distance. At the outset any inaccuracy in the collimation of the transit-instrument placed over the pit mouth would give erroneous points at the bottom, and again in placing the instrument at the bottom of the pit, if its axis was not fairly over the point registered in sighting down the pit, whatever the deviation might be here, on this short base-line, the inaccuracy would become larger and larger as the survey advanced. It must be admitted that on plain traversing work, and using the greatest care, the imperceptible differences in reading the verniers would amount to a few minutes even in a small survey, but in this case there is more chance of such small errors compensating each other than in sighting down the pit, and the building up or construction of a large plan on so short a base-line. In preference to using the transit-instrument for sighting down the pit and using a base-line of such limited length, he thought much better results would follow by making a careful traverse, and triangulating between two or more pits, and the more pits connected with the surface the better, because they best of all prove the accuracy or the contrary of any underground survey. In this case the base-line might be of very considerable length, and all that was required was that the centre of the shaft shall be carefully noted in the measurement from the starting point. On the centre of the second shaft being reached its centre would also be carefully noted, and, if possible, the survey continued in a circuit back to the starting point of the first line and the angle taken upon it to its end. On the completion of such a circuit, any error in the triangulation would show on the last reading of the verniers, or could be proved by summing up the whole of the interior angles.* The shafts being vertical, the centres at the tops will be over the centres at the bottoms of the shafts, and on completion of the plan plotted from such a survey as that named if the centres of the shafts as shown by it be placed on a plan of the surface drawn to the same scale, both being accurate they will coincide with each other ; and similarly equal care having been used the whole of the workings so surveyed will be shown in their true position in reference to the boundaries or any other objects on the surface of the ground. If it be not practicable to traverse the workings around the bottoms of the pits, the survey ought to be made back over from No. 2 pit to the starting point of the survey or of the first line measured.

* All the interior angles of an irregular polygon are equal to twice as many right-angles as the polygon has sides, less four right-angles, and any difference would be the amount of error.

By fixing permanently the points of such surveys, every line between any two points practically becomes a base-line for future use. He (Mr. Ramsay) further remarked that where it was possible to traverse a circuit round the bottoms of any number of pits it was desirable to do so, because the imperceptible errors arising in reading the vernier and the adjustment of even the best instruments might compensate each other, or at all events they would be minimised.

The PRESIDENT said that Mr. Henderson, in his paper, mentioned one or two points upon which Mr. Beanlands might say a few words. One of them was "in the now abandoned Crown portion of this mine, which extends for a long distance beneath the bed of the Atlantic Ocean, a great amount of local magnetic attraction prevails, rendering compass-bearings quite unreliable."* Then he went further in the same direction as Mr. Ramsay: "In a recent underground survey at Wheal Cock, the surveyor was obliged to seek for a true bearing at a distance of ¼ mile from the shaft, and using a draft-line at that place of only 16 feet in length as his base, he carried on the whole of his subsequent survey by means of an angular instrument of recent invention, without reference to any further needle-reading. The work involved no less than 363 draft measurements, and occupied seven days in its accomplishment."† The shaft might be of such a character that they could not look down it from the surface to the bottom, and what he was anxious to ascertain was what steps might fairly be taken by any one who had to bring into account the variation owing (it might be) to local circumstances, or the ordinary magnetic variation, to ensure correctness.

Mr. BEANLANDS said that he was not sure whether he understood the question aright. In cases where the shaft was not perpendicular, and they could not see from the top to the bottom, he knew of no better method than of either surveying to a second shaft, if there was one within a reasonable distance, or falling back upon the miner's compass. With regard to the width of the shaft, he had made the necessary connection in many shafts of different widths, but had rarely had occasion to make a survey where the shaft was not of sufficient width for the purpose. Several years ago a bearing was taken at Ryhope colliery in a shaft sunk to the depth of 1,524 feet. The process was repeated a year or two afterwards, the two results being nearly identical. If two independent plans had been made from the two bearings the difference would not have been perceptible at a distance of two miles from the shaft. He had had considerable experience in surveying from two

* *Trans. Fed. Inst.*, vol. viii., page 273. † *Ibid.*, page 274.

shafts. In one case the shaft was inconvenient for planting the transit-instrument, owing to the presence of timber of a character not suitable for mounting the instrument, and he adopted a base-line from two shafts. There is occasionally some difficulty and great extra labour involved in surveying from two shafts, and the results are rarely as accurate as those obtained from the transit-instrument.

Mr. J. A. RAMSAY agreed that it was quite impossible to get a base-line by hanging strings down the pit. Certainly the proper method of surveying must be to get as nearly as possible point over point. In all surveying there were so-called imperceptible errors arising from the reading of the instruments, and in taking a transit-line down a shaft up to 14 feet in diameter, the base-line was too short. In making mining surveys the magnetic base-line of the miner's compass ought not to be used excepting only as a continuation of the minor workings from the lines and points established by the theodolite.

Prof. STROUD wrote that he thought the magnetic compass should continue to prove of important service in mine surveying. The periodic variations (viz., annual and diurnal) were too small to affect its ordinary use. The secular variation, although small, was cumulative, and should of course be taken into account. The irregular variations were known from the records taken at magnetic observatories, as wide areas were affected simultaneously. The local disturbances noted by Profs. Rücker and Thorpe were not serious in Northumberland and Durham. He might mention that the magnetic axis was not necessarily coincident with the line joining the ends of the needle, and perhaps some of the difficulties experienced in surveying might be due to this cause. In case the magnetic axis was not coincident with the line joining the ends of the needle, the magnetic meridian would not be exactly given by a single reading of the needle. The error, however, might be avoided by taking a second reading with the other face of the needle turned downwards, and the mean of the two readings would give the correct magnetic meridian. His (Prof. Stroud's) remarks on this head were rather to suggest that the magnetic meridian for very careful work could not necessarily be obtained with a single reading. In making an underground survey, no definite conclusion could be drawn as to the magnetic meridian unless the needle remained parallel to itself, as tested at several positions along a base-line. If the direction of the magnetic needle be indicated on the plan at the various positions of the survey, and it be found that the needle remained parallel to itself at several positions, as tested along a base-line, then it might be concluded that there was no

magnetic disturbing cause in the immediate neighbourhood. He (Prof. Stroud) considered that no conclusion could be drawn from the compass unless an undisturbed position could be found somewhere in the workings.

Mr. JAMES HENDERSON (Truro) wrote that the remarks made during the discussion rendered it desirable that he should enlarge somewhat on his previous statements, with the object of rendering them of greater service to the mine surveyor. It was said that the compass although possessing "considerable advantages for underground surveying was not, however, an infallible instrument, particularly in mines where much iron was used," and that its use "was not attended with that degree of precision which would be required in special surveys such as the defining of boundaries, etc." The author differs widely from this expressed opinion: From his experience during many years of the use of the miner's dial or compass in Cornish metalliferous mines, where, except perhaps in the instance of the Crown shaft of the Botallack mine, no underground survey could be connected by means of a transit or theodolite with the surface as is the case in collieries, he firmly believed in the adaptability of the magnetic needle for the minutest survey, even where inches are of importance, as they would be in the boundary of a valuable mine, but then it must be used with care and circumspection throughout, and bearings only recorded where proof could be obtained of their correctness. Any traverse over rails or in the neighbourhood of local attraction must be made either by means of what is termed the rack in the miner's dial, with the needle thrown off, or by entering the bearing of the loose needle for every back and forward sight, and afterwards calculating the correct bearing of each line of the traverse. The adoption of this system he (Mr. Henderson) highly recommended, especially where the tripod-stands used are old and shaky and liable to the effects of torsion. Where the legs are firm and rigid any angular instrument of proper construction may be used where local attraction occurs, provided one or more drafts of the traverse be entered at their true magnetic bearing. He (Mr. Henderson) had found his "rapid traverser" well adapted for this purpose, the liability to error in reading and booking the angles or bearings of the traverse drafts being entirely avoided by its use.* It had been stated during the discussion that the miner's compass could not be read to less than one quarter or one eighth of a degree. He (Mr. Henderson) had always used in his miner's dial a needle mounted on a ruby, with a vernier attached to its north-seeking end reading to single minutes, and very precise bearings could in

* *Trans. Fed. Inst.*, vol. v., page 199.

consequence be taken by its means. It was highly important that both surface and underground surveys should be made with the same instrument if it be necessary to connect them. There was nothing gained by surveying at the surface with a costly transit or theodolite and attempting the underground work with a humbler instrument. On the contrary, in all probability their readings would differ considerably, and there would be a corresponding error in the resulting plan. Whereas, by working with the same instrument in both instances, even if the less showy one be used, the traverse would most likely be found to be correctly performed.

DISCUSSION ON MR. J. D. KENDALL'S PAPER ON "MISS-FIRES." *

Mr. J. D. KENDALL wrote that the following Special Rule had been adopted in certain hematite mines under his charge :—

All holes fired in this mine must be charged in such a way that not more than 5 inches of stemming shall be used in holes 18 inches deep and under, and not more than 8 inches in a hole of greater depth than 18 inches. In case of a miss-fire, no attempt shall be made to remove the stemming, but another cartridge must be placed on the top of it and one or two inches of additional stemming. The latter cartridge must then be fired in the usual way. If the bottom cartridges should not be fired by this means, the hole must be left and information given at once to the manager or his deputy. Further, in case of a miss-fire, no one shall enter the working where such miss-fire has occurred before the lapse of twenty minutes.

DISCUSSION UPON MR. R. RUSSELL'S PAPER ON "THE EXTENSION OF THE WEST CUMBERLAND COAL-FIELD, ETC."†

Mr. J. D. KENDALL (Whitehaven) wrote that he found that some of the details in Mr. Russell's paper could not receive general acceptance. Mr. Russell said that "the workings from the old shafts in the vicinity of Preston Hows were carried under the St. Bees Sandstone. From the Croft pit, the coal had been worked for some distance in the direction of Sandwith, and the base of the red rock proved by boring upward from the underground workings."‡ The red rock here referred to was clearly supposed to be St. Bees Sandstone. That view had also been held by the colliery officials and, in consequence, the 240 feet fault or dyke beyond which the red rock was proved had, since 1849, when the boring was made, been known as the St. Bees dyke. Although not material to the subject

* *Trans. Fed. Inst.*, vol. vii., page 605. † *Ibid.*, vol. viii., page 276. ‡ *Ibid.*, page 278.

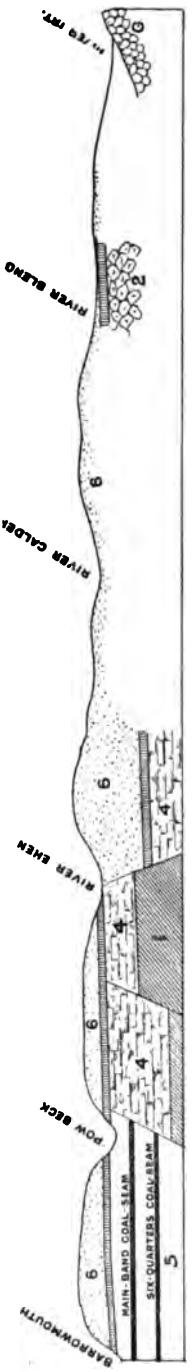


Fig. 4.

of Mr. Russell's paper, it might be stated, as a matter of fact, that the red rock in question was not St. Bees Sandstone but Whitehaven Sandstone, as would appear from a consideration of the evidence. The boring was put upward from the yard band drift, at a distance of about 660 feet from the fault and on the downcast side of it. The rocks passed through were the ordinary shales, sandstones, and coals of the Middle Coal-measures up to 173 feet in thickness. Red sandstone was then entered for 9 feet until the boring was stopped. Before the St. Bees Sandstone could have been reached the hole would have had to pass through about 167 feet more of Whitehaven Sandstone and then through a greater or less thickness of breccia, Magnesian Limestone, and red shales with gypsum, as there was no fault which could have thrown off these rocks. Mr. Russell further says that "the inclination of the coal-seams at Whitehaven is towards the west and south of west, and the direction of the dip of the limestone in the neighbourhood of Cleator and Egremont has a similar bearing." And further, "taking the south-westerly direction of the dip as continuous throughout this comparatively unknown district to the south of Egremont, . . . the strike of the measures would range somewhat in the direction of the thick broken line shown on the map and approximately parallel to the coast-line."* The sections shown by Figs. 2 and 3 (Plate XVII.), it will be seen on reference to Fig. 1 (Plate XVI.) are parallel to one another and to the strike of the known and supposed coal-fields. The coal-seams appearing in Figs. 2 and 3 should therefore have been level and not dipping northward as shown. The same remark applied to the Carboniferous Limestone. There was not sufficient data to enable anyone to make a section

Fig. 4.—Section from Barrowmouth to Muncaster Fell.

References.

1. Skiddaw slate.
2. Volcanic series of Borrowdale.
4. Carboniferous Limestone.
5. Coal-measures.
6. St. Bees Sandstone.
G. Granite.

* *Trans. Fed. Inst.*, vol. viii., page 279.

along the line of Fig. 3, but on the line of Fig. 2 there was a considerable amount of information, but this Mr. Russell appeared to have disregarded, or his section would have been similar to Fig. 4. Mr. Russell also stated that the probable depth of the Coal-measures would be from 1,300 to 1,400 feet.* There was not a scrap of evidence to support this statement, nor for other similar estimates of depth near Allonby and Carlisle; indeed, in the present state of knowledge such estimates were the merest guesses, and more information must be obtained before any reliable opinion can be expressed.

DISCUSSION ON MR. SIMON TATE'S PAPER ON THE 'SAVING OF LIFE FROM AFTER-DAMP, SMOKE, OR FUMES IN MINES."†

Mr. TATE said that he had had an opportunity of testing his proposed system in a small district, which was shut off for twenty hours from the ventilation of the mine. After this interval, on the district being re-entered, they were able to visit the greater part of the district, and to reach the face of the workings. This test showed that his proposals were capable of being of practical use.

Mr. W. RAMSAY (Tursdale) said that all mines were not so situated that the doors contemplated by Mr. Tate could be inserted. He also suggested that the doors might not always be in working order when an explosion took place, even if the men were properly instructed how to shut the doors. Men should be instructed to open the separation-doors so that the after-damp of an explosion could get away into the return-airways. If the mine was ventilated by a furnace or furnaces, and the fire was blown off the furnaces, the heat remaining in the shaft would maintain the ventilating-current. If the ventilation was produced by means of a fan, it would be necessary to run it as hard as possible, so as to get the after-damp out of the pit. If a fire was found on any of the roadways, the mode of procedure would depend on circumstances; if it was simply burning coal, he would think nothing of passing beyond it; but if it was inflammable material, every precaution should be used. He thought all officials, especially deputies and overmen, should carefully consider what should be done if an explosion should take place at the out-bye end of the mine.

* *Trans. Fed. Inst.*, vol. viii., page 280. † *Ibid.*, page 189.

Mr. J. A. RAMSAY (Littletown) said that the essential thing to be done after an explosion was to get the ventilation restored. He suggested the use of duplicate separation-doors, which could be put into use should those ordinarily employed be destroyed. He thought it undesirable that there should be a long drift on the surface between the top of the upcast shaft and the fan. He considered that it was much safer to place the fan within a reasonable distance, say 8 or 10 feet, from the top of the upcast shaft. If an explosion occurred, and its effects reached to the surface, the force of the explosion would take a direct line upwards, leaving the fan undamaged. The drift connecting the fan with the top of the upcast shaft would be exposed to less risk of damage or destruction, and from its short length of 10 or 15 feet would be more speedily repaired than a longer length.

Mr. S. TATE said that some of the members suggested that it was only necessary to open a door here or to break a hole there, in order to remove the after-damp; but more than that was required in order to prevent the after-damp from passing into some of the districts which had not been affected by the explosion. In one case, the lives of 126 men were saved through a deputy placing them between two ventilating-doors. The deputy went in-bye and found the after-damp, a crossing having been blown out on the out-bye side. He took his men back to the ventilating-doors, and all of them were saved. Many men had been killed after an explosion, as they were not properly acquainted with the return-airways or did not know what to do.

The PRESIDENT said no doubt Mr. Tate had instructed the principal officials at the collieries under his charge, as to the course they should adopt on the occurrence of an explosion; and possibly when he had reduced his rules to writing, he might give the members the benefit of a knowledge of these rules. His paper was a very valuable one, and pointed to the desirability of increasing the knowledge of the officials in charge of collieries, so that they might use a large amount of intelligence in the event of any explosion taking place.

Mr. J. L. HEDLEY (H.M. Inspector of Mines) said he could not allow the opinion to pass unchallenged as to the proper position of a ventilating-fan. He did not know whether Mr. Ramsay was speaking from actual experience as to the results of an explosion or not, but it would be well if he (Mr. Hedley) were to quote an occasion where the fan was some distance from the upcast shaft; and owing to its being at such a distance it was found undamaged after an explosion, and was capable of use immediately afterwards. Speaking in a general way, he was of opinion

that the farther away the fan was placed from the top of the upcast shaft the less liable it was to injury.

Mr. J. A. RAMSAY said that he believed the right course to adopt was to place the fan closely adjacent to the top of the upcast shaft, which should be covered by a light valve, so that the force of the blast of an explosion might not be diverted into the fan-drift and cause serious damage to the drift or to the fan.

Mr. J. L. HEDLEY said that at Abram colliery, in Lancashire, where 50 or 60 lives were lost by an explosion, the Guibal fan resumed work within a few minutes of the accident. The top of the upcast shaft and the top of the drift from the shaft to the fan were destroyed, but were temporarily replaced. If this fan had been placed within a few feet, instead of 120 feet, from the top of the upcast shaft, it would certainly have been destroyed. The Guibal fan, placed 150 feet from the upcast shaft at Pemberton colliery, near Wigan, was uninjured, and was not even stopped by the explosian.

Mr. T. BELL (H.M. Inspector of Mines) said that it was a fact well known to all practical mining engineers that, if the fan was placed within a few feet of the upcast shaft, it would certainly be damaged by the blast of an explosion. In addition, General Rule 3, of the Coal Mines Regulation Act of 1887, provided that a fan should be placed at a sufficient distance from the pit to prevent its being injured by an explosion.

Mr. T. W. BENSON (Newcastle-upon-Tyne) said that he was glad to be fortified by the opinion of two of H.M. Inspectors of Mines, that he had not been wasting about 50 per cent. of the expense of a new fan by placing it at a distance of 60 feet from the top of the upcast shaft.

Mr. T. L. ELWEN read the following paper on "The Effect of an Obstruction in the Airway of a Mine":—

THE EFFECT OF AN OBSTRUCTION IN THE AIR-WAY OF A MINE.

By T. L. ELWEN.

The subject-matter of this paper, based on a statement made by Mr. D. Murgue in his paper on "The Friction of, or Resistance to, Air-currents in Mines,"* ought more properly to have been brought forward in the discussion which followed. A protuberance of the roof is shown in Fig. 22, with reference to which Mr. Murgue states "If this protuberance were preceded and followed by a progressive curvature of the roof (as shown by the dotted line) corresponding to the depression of the seam, the amount of pressure absorbed in overcoming the obstacle would be restored after passing the obstacle, and there would be no loss. But the sudden widening beyond the protuberance causes a swirling and eddying, and a loss of pressure which is measured by the air-column due to the loss of velocity." However, in view of the importance of the point raised, further experiments seemed necessary, which are herewith recorded.

The position chosen for the experiments was a main intake haulage-road, of which Fig. 1 is a plan, and Fig. 2 a cross-section. The figures show the position of the obstruction and other bratticing erected for the purpose of the experiments.

The air-measurements were made by dividing the air-way by strings into squares, and the velocity was taken separately in each. These measurements were made at points at short distances out-bye and in-bye from the bratticing. Very little difference was noted between the two measurements, and the mean quantities are recorded in the tables.

Other necessary precautions and arrangements were observed to ensure accuracy. In each experiment, the velocity of the air-current was also observed at the water-gauge and end of pipe, as a correction in working out the differences of pressure (see Figs. 1 and 2).

The water-gauge F was screwed to a stand in the centre of the air-way, and near the commencement of the bratticing. Small iron pipes were attached to the water-gauge and were carried forward and terminated near the end of the bratticing G in the centre of the air-way. The end of the pipe was "padded." A range of hose-pipe was also connected with the

* *Trans. Fed. Inst.*, vol. vii., page 212, and Fig. 22, Plate VIII.

water-gauge with arrangements for making a connexion when necessary, and terminated in the centre of the open space at the obstruction *C*. Thus the water-gauge readings from the iron pipes represents the difference of pressure between the points *A B*, and is designated "pipe water-gauge." The second readings, designated "hose water-gauge," represents the difference of pressures existing between the point *A* and the obstruction *C*. The total length of air-way *A B*, covered by the experiments, is 84 feet, with an average sectional area of 34·375 square feet, perimeter of 23·5 feet, and total rubbing-surface of 1,974 square feet. At the obstruction, the sectional area is reduced from 30 square feet to 6·78 square feet.

Experiment No. 1.

The obstruction and the tail-brattice *D* and *E* at each end having been erected, the following observations were made:—Volume of air per minute, 25,545 cubic feet; velocity of air-current per minute at water-gauge, 990 feet; velocity of air-current per minute at pipe-end, 770 feet; pipe water-gauge, 0·465 inch; hose water-gauge, 1·000 inch.

Experiment No. 2.

In this experiment advantage was taken of adjoining ventilating-doors to divert part of the air-current. The object being to ascertain the effect of a reduction of volume under conditions similar to those of experiment No. 1:—Volume of air per minute, 15,444 cubic feet; velocity of air-current per minute at water-gauge, 600 feet; velocity of

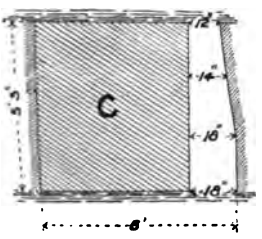


Fig. 2.—Section at Obstruction *C*.
Scale, 6 feet to 1 inch.

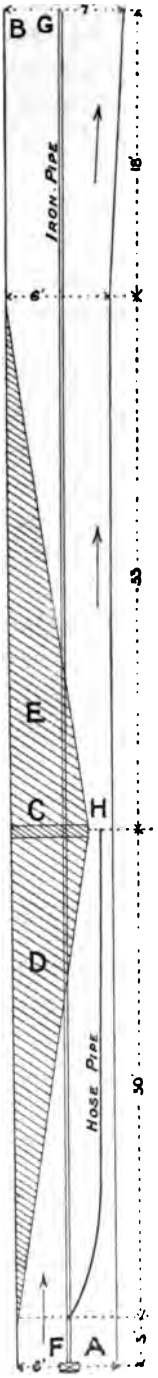


Fig. 1.

Fig 1.—Plan.
Scale, 12 feet to 1 inch.

References.

—→ Direction of air-current.
C.—Obstruction.
D.—Outbye brattice.
E.—Inbye brattice.
F.—Water-gauge.
FG.—Iron pipe.
FH.—Hose pipe.

air-current per minute at pipe-end, 470 feet; pipe water-gauge, 0·175 inch; hose water-gauge, 0·375 inch. With similar ventilating arrangements and with the whole of the brattice and obstruction removed, the quantity of air per minute was 17,429 cubic feet.

Experiment No. 3.

The ventilation in this case was on its normal course, or under conditions similar to those prevailing in experiment No. 1. The out-bye end of the brattice *D* was taken down, the obstruction *C* and brattice *E* remaining in position:—Quantity of air per minute, 24,570 cubic feet; velocity of air-current per minute at water-gauge, 990 feet; velocity of air-current per minute at pipe-end, 820 feet; pipe water-gauge, 0·500 inch; hose water-gauge, 0·900 inch.

Experiment No. 4.

The in-bye end of the brattice *E* was taken down, leaving the obstruction *C* only in position:—Volume of air per minute, 22,308 cubic feet; velocity of air-current per minute at water-gauge, 920 feet; velocity of air-current per minute at pipe-end, 710 feet; pipe water-gauge, 0·700 inch; hose water-gauge, 0·725 inch.

Experiment No. 5.

To ascertain what differences, if any, existed in having the velocity of the air-current gradually brought up to, and the swirling and eddying as it leaves the obstruction, the out-bye end of the brattice *D* was re-erected, and the obstruction *C* left in position:—Volume of air per minute, 23,049 cubic feet; velocity of air-current per minute at water-gauge, 990 feet; velocity of air-current per minute at pipe-end, 740 feet; pipe water-gauge, 0·600 inch; hose water-gauge, 0·975 inch.

Experiment No. 6.

As a correction in working out the pressures, it was necessary to ascertain the amount of pressure lost on the rubbing-surface. For this purpose the whole of the obstruction *C* and brattices *D* and *E* were removed; the length of the air-way was extended to 205 feet, and it had a rubbing-surface of 4,817 square feet:—Volume of air per minute, 27,495 cubic feet; velocity of air-current per minute at water-gauge, 1,210 feet; velocity of air-current per minute at pipe-end, 1,030 feet; pipe water-gauge, 0·0468 inch.

From the data of No. 6 experiment, one can ascertain the loss of pressure and the co-efficient of friction for the rubbing-surface of the air-way. Thus the average velocity $= \frac{27,495}{34{\cdot}375} = 800$ feet per minute.

The difference of pressure $= 0{\cdot}0468 \times 5{\cdot}2 = 0{\cdot}243$ pound per square foot. The correction for difference of velocity

$$0{\cdot}243 + \left[\left(\frac{20{\cdot}17^2}{64{\cdot}4 \times 13}\right) - \left(\frac{17{\cdot}17^2}{64{\cdot}4 \times 13}\right)\right] = 0{\cdot}377 \text{ pound per square foot.}$$

The correction for the standard velocity of 1,000 feet per minute $\frac{0{\cdot}377}{0{\cdot}8^2} = 0{\cdot}589$ pound per square foot.

The co-efficient of friction $\left(= \frac{0{\cdot}589 \times 34{\cdot}375}{4,817} = \right) 0{\cdot}0042$ pound per square foot, practically corresponds with the value obtained by the writer in previous experiments on the same kind of air-way.*

The loss of pressure, as ascertained by the above calculation, due to the rubbing-surface in each experiment, was deduced from the area, velocity, etc., for each length of 10 feet from *A* to *B* when all the brattice or part, and when the obstruction only was in position. The value of the co-efficient of friction was used throughout, although the sides of the brattice as a plain rubbing-surface would offer rather less resistance.

TABLE I.—SUMMARY OF RESULTS OF EXPERIMENTS.

No. of Experiment.	Conditions of Experiment.	Volume of Air per Minute.	Pipe Water-gauge per Square Foot.	Velocity of Air-current at Water-gauge per Minute.	Velocity of Air-current at Pipe-end per Minute.	Corrected Water-gauge for Difference of Velocity per Square Foot.	Current Resistance per Square Foot.	Corrected Water-gauge, the Current Resistance being Deducted, per Square Foot.	Proportion of Current Resistance to Total Resistance.
		Cubic Ft.	Lbs.	Feet.	Feet.	Lbs.	Lbs.	Lbs.	
1	Obstruction C and brattice D and E	25,545	2·420	990	770	2·550	0·970	1·580	0·38
2	Do. do.	*15,444	0·910	600	470	0·950	0·350	0·600	0·37
3	Obstruction C and brattice E ...	24,570	2·600	990	820	2·700	0·670	2·030	0·25
4	Obstruction C ...	22,308	3·640	920	710	3·750	0·380	3·370	0·10
5	Obstruction C and brattice D ...	23,049	3·120	990	740	3·260	0·670	2·590	0·20
6	No obstructions ...	27,495	0·243	1,210	1,030	0·377	0·004	0·373	—
		(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)

* With all obstructions removed the volume was 17,429 cubic feet per minute.

Taking the conditions prevailing in experiment No. 1 as a basis, the following results may be noted:—

* *Trans. N. E. Inst.*, vol. xxxviii., page 215.

TABLE II.

No. of Experiment.	Conditions of Experiment.	Decrease of Air.	Increase of Pressure.	Reduction of Volume of Air, the open Air-way being taken as a Basis.
		Per Cent.	Per Cent.	Per Cent.
1	Obstruction C and brattice D & E	Ba	sis.	7
2	Do. do.	—	—	11
3	Obstruction C and brattice E ...	4	28	11
4	Obstruction C	13	113	19
5	Obstruction C and brattice D ...	10	64	16

Other results are recorded in Table III.:—

TABLE III.

No. of Experiment.	Quantity of Air per Minute.	Average Velocity per Minute at Obstruction.	Highest Velocity per Minute at Obstruction.	Co-efficient of *Vena Contracta*.
	Cubic Feet.	Feet.	Feet.	
1	25,545	3,768	4,936	0·76
2	15,444	2,278	3,014	0·75
3	24,570	3,624	4,734	0·77
4	22,308	3,290	4,280	0·77
5	23,049	3,400	4,884	0·69
(1)	(2)	(3)	(4)	(5)

The velocity stated in column 4 of Table III. is found by adding together the velocity of air-current at the water-gauge and the velocity due to the head or equal hose water-gauge. It will be observed that the co-efficient of the *vena contracta* is nearly constant in all the experiments, except the last, in which case the brattice *D* was taken down, and, as might be expected, the co-efficient is smaller.

Conclusions.—So far, then, as the full brattice and obstruction is concerned, instead of "the loss of pressure being again restored" there is a definite reduction of pressure. This is further proved by the reduction of the quantity of air in two experiments, equal to 7 and 11 per cent. respectively. After deducting the loss of pressure due to the current-resistance calculated by using the co-efficient of friction, as previously shown, there still remains a further loss of pressure to be accounted for. (See experiment No. 1, Table I., columns 7 and 8.) So far as the writer can see, this can only be accounted for by the fact of the air-current, tending to move in a straight line, being constantly deflected from a straight course at the brattice-sides from the point *A* to the obstruction *C*, and producing an additional resistance to the air-current.

It is evident that there is less resistance produced by increasing the speed of the air-current gradually up to the obstruction, than by gradually reducing the velocity after it leaves the obstruction. With the obstruction only, the loss of pressure is practically equal to the head required to produce the velocity at that point, a fact which confirms the accepted theory.

The PRESIDENT moved that a vote of thanks be accorded to Mr. Elwen for his paper, and the discussion was adjourned.

Mr. W. O. WOOD read the following paper on "The Murton Coal-washer":—

THE MURTON COAL-WASHER.

By W. O. WOOD.

It is generally admitted that the trough-washer is the simplest and the most efficient method of washing coal, so far as the purity of the products is concerned. But unfortunately it is expensive, in consequence of the excessive labour required, and wasteful, in consequence of the large amount of fine coal which is unavoidably carried away amongst the washed-out dirt. It has been generally discarded in favour of mechanical systems, which, perhaps, may not be so efficient, but are less wasteful and less expensive to work.

It is clear that if a machine could be contrived in which all the merits of the trough-system could be retained, and its defects remedied, a perfect washer would be secured. This has been attempted by the inventors of various machines, but with only partial success.

The Murton coal-washer is a trough or stream-washer, which perfectly fulfils these conditions. It combines the simplicity and efficiency of the trough-system with perfect washing, absence of waste, and economy of labour. It has now been in successful operation for upwards of one and a half years, and 400 tons of very fine duff-coal are treated per day of ten hours, with most satisfactory results.

The apparatus, which has been named the Murton coal-washer, consists of an endless articulated steel trough-belt, which is watertight. This trough revolves slowly round suitable drums at each end, to which motion is imparted by a simple arrangement.

The washer is continuous and automatic in its action, the clean washed coal falling into the washed coal-hopper at the lower end, and the refuse or dirt being delivered into a dirt-hopper at the upper end. The coal is washed perfectly clean and free from impurity, while the dirt is delivered practically free from coal. The cost of labour is reduced to a minimum, the same number of workpeople being required, as in all other systems of mechanical coal-washing.

The washer is equally available for all grades of coal, and is efficient for the finest and smallest as well as for the larger sizes. It will also wash mixed coals when required. For treating very small and fine coal it is, in the experience of the writer, the best system now in use.

The accompanying plans show the details of the arrangement. Fig. 1 is a general view of the plant; Figs. 2, 3, and 4 show the details of the trough-belt; and Fig. 5 shows the arrangement for securing a watertight joint (Plate II.).

Description of the Washer.

The steel trough is 60 feet long, 3 feet wide, and 8 inches deep, and it is fixed at an inclination of about 1 in 18. The supporting drums and rollers are mounted on a suitable frame. Dams or stops about 2 inches high are placed at intervals of 3 feet inside the trough. The joints are so constructed as to be perfectly watertight without the use of packing, and the dams form half of the joint of each 3 feet tray.

The method of feeding the coal into the trough is most effective in the prompt separation of the denser impurities from the coal. A short inclined plate (designated the "feeding-plate") is arranged immediately below the water-nozzle, above which is the coal-supply spout. The water in flowing down the feeding-plate floats and carries the dry coal, and delivers it with considerable force into the trough. The greater portion of the dirt is immediately precipitated and carried by the moving trough to the dirt-hopper, the coal flowing down the trough with the water. By this arrangement only 12 feet length of trough is required for washing away the dirt, thus allowing a length of 48 feet to be used for coal-washing.

The water is supplied by arrangements of the usual character, and the same water is used over and over again. About 450 gallons of water per minute are circulated when washing 400 tons of coal per day.

Method of Working.

The trough is set in motion and travels up the incline, towards the coal and water-supply, at a speed of from 8 to 10 feet per minute. The coal from the hopper *H* and the water through the nozzle *E* are turned on in suitable proportions, and the water, carrying with it the coal, flows downwards in a direction contrary to the travel of the trough, to the coal delivery end *D*, where the coal is delivered free from impurities.

The dirt, falling to the bottom of the trough, is arrested by the dams or stops *C C*, and is carried forward by the motion of the trough-belt beyond *E*, the point of supply. As soon as it passes this point it is agitated, so that any coal that is mixed with it is liberated and carried by the strong current of water supplied by the nozzle *J* downwards to *E*, where it joins the regular coal-feed, and is carried downwards with the other clean coal to *D*.

The refuse continues to be carried upwards to the end of the trough *M*, and as the belt turns round the drum *A*, it falls off into the hopper. The trough is washed clean by the water-spray *S*, and passes to the drum *B* at the lower end *D*.

The coal when delivered at the lower end *D* falls into a spout lined with a draining-plate (with 125 perforations of $\frac{1}{16}$ inch diameter per square inch), and the water is quickly drained away. The coal slides down the spout into the elevator-boot, whence it is raised and delivered into a large hopper, from which it is run into the coke-oven tubs as required.

In consequence of the very fine character of the coal, the drainage-water is heavily charged with it, and it has been found advisable to run the drainage-water through a series of settling-tanks, where part of the fine coal is deposited. After passing through these tanks the water returns to the pumps, and is used over again.

When the settling-tanks are filled (they are four in number and two are used alternately) the fine-coal or sludge is run out through the slides at the bottom on to a creeper-belt which conveys it to the boot of the elevator, where it is mixed with the other coal. The recovery of the sludge is perfect and inexpensive, there is no waste of water, and no fouling of the neighbouring streams.

All the washed coal is made into coke.

Cost of Washing.

The washery is worked as a separate establishment, and the coal is conveyed to it from the various pits in waggons. One workman is employed to tip the waggons, and another to attend to the engine and boilers; these workmen would not have been required if the washer had been attached to the pit-heap, the coal delivered or elevated direct from the small-coal apparatus, and power available from the other plant.

The cost of washing 400 tons per day, including all charges, is 0·76d. per ton. If the plant had been attached to the pit-heap the cost would have been 0·48d. per ton on the coal washed.

The washed coal contains 3·71 per cent. of ash, and the coke made from it contains 5·39 per cent. of ash. The coke made from unwashed coal contains 13·28 per cent. of ash.

It will be observed that this coal-washer is of an exceedingly simple and inexpensive construction, and the results, particularly in treating fine coal, are equal in every respect to what is obtained from much more expensive and elaborate plant.

Patented "pal-washer."

VOL IX. PLATE II

FIG. 5.

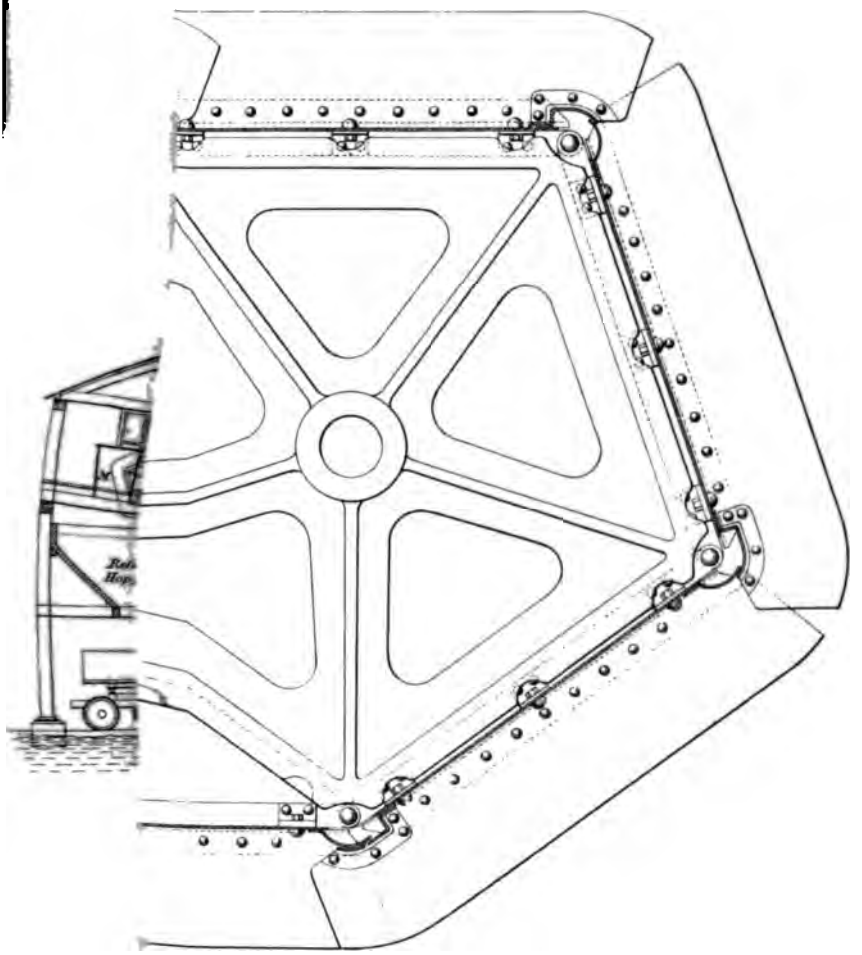
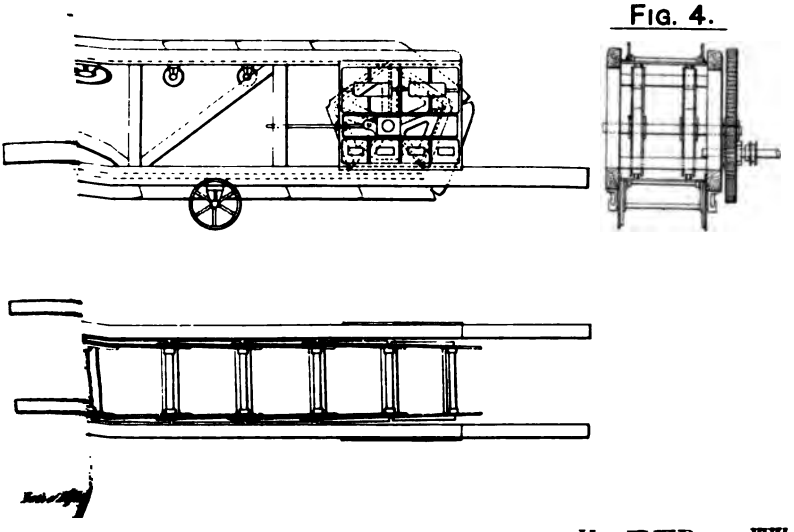


FIG. 4.



VOL XLIV. PLATE XVII.

In consequence of the slow speed of the machine, the cost of maintenance is small. The machines, at Murton, have not required repair since they started one and a half years ago, and they do not appear likely to require any repairs for some time to come. The power required to drive the trough-belt, apart from that used in pumping the water, which may be taken to be the same in every system, is very small, say, for two machines, about 1½ horse-power.

Mr. T. E. FORSTER (Newcastle-upon-Tyne) enquired as to the nature and percentage of the dirt washed out of the coal?

Mr. W. COCHRANE (Newcastle-upon-Tyne) said that he had seen the coal-washer in operation, and he considered that Mr. Wood had not overstated the delivery of coal or the dirt washed out. Perhaps Mr. Wood would give the actual cost of the Murton coal-washing plant for an output, say, of 400 tons per day, without any extraordinary arrangements for handling the coals.

Mr. W. C. BLACKETT asked what was the size of the mesh over which the "fine duff coal" was passed? As to the comparison with more elaborate plants, such as the Lührig washer, there was a good deal of misapprehension, and if the Lührig were examined in comparison with other coal-washers he thought that they would not be found to be so much more expensive. The large sums so frequently referred to were paid for complete erections, including screening-belts, screens, settling-tanks, pumps, elevators, etc., in fact, the entire plant placed and used at the pit's mouth till the washed coal was delivered. If the Robinson, the Murton, and other coal-washers were attached to complete surface-plants, he thought that they would bear a similarly high cost.

Mr. F. BERKLEY asked what percentage of coal was contained in the refuse?

Mr. R. THOMPSON asked whether the sludge found in the settling-tanks had been analysed?

Mr. W. RAMSAY asked what percentage of sulphur was contained in the coke?

Mr. W. O. WOOD said that, at Murton colliery, the coals from three seams were washed and, as might be expected, the refuse was of a mingled character. One coal-seam contained much fine dirt, iron-pyrites, and shales; and another contained a friable band, but no difficulty was found in separating the refuse. There was the ordinary difference between the specific gravity of the coal and the refuse. The quantity of dirt washed

out was about 16 or 17 per cent. of the coal washed. If the washer were simply adapted to a pit-heap, as most washers were, the cost would be very little; but owing to the geographical position at Murton, the coals required to be elevated to the washer and from the delivery-spout of the washer to the coal-hopper supplying the coke-ovens. The cost of the whole plant, including the settling-tanks, which were of an inexpensive character, was about £2,000. The settling-tanks were not absolutely necessary, but they were used at Murton as a matter of convenience, so as to avoid any risk of the pipes or of the pump being choked. The gauze through which the duff coal passed (it was no guide or indication of the size of the coal going through, as the coal was of a very dusty character) was of $\frac{3}{8}$ inch mesh. The percentage of coal in the dirt did not appear to exceed 1 per cent. of the quantity of dirt. The sludge had not been analysed. The percentage of sulphur in the coke had been decreased, but it was one of those things that could not be washed out of the coal.

The PRESIDENT moved that a vote of thanks be accorded to Mr. Wood for his paper, and the discussion was adjourned.

The following paper by Dr. R. Brunck on "The Brunck Coke-oven and System for the Recovery of Bye-products" was taken as read :—

THE BRUNCK COKE-OVEN AND SYSTEM FOR THE RECOVERY OF BYE-PRODUCTS.

By R. BRUNCK, Ph.D.

The plant for coking with recovery of bye-products described in this paper has been erected and has been worked on an experimental scale at the Kaiserstuhl colliery, near Dortmund, and is the invention of Mr. F. Brunck.

A trial-battery of six ovens has been working since September, 1893, and is provided with a separate condensing-plant for the recovery of the bye-products—tar, sulphate of ammonia, and benzol.

The main features in the construction of the ovens (Figs. 1, 2, 3, 4, and 5) are :—

(1) The oven-chambers are 32·8 feet (10 metres) long, 6·56 feet (2 metres) high, and from 15 to 21½ inches (38 to 55 centimetres) wide in the middle (according to the quality of coal), and have vertical wall-flues like the Coppée-Otto ovens.

(2) In each wall, between two ovens, there are two series of heating-flues, separated by a solid wall carrying the entire weight of the vault. The heating-walls, having no weight to support, can consequently be constructed as thin as possible, so as to obtain the highest degree of heating effect.

(3) This method of separation provides each oven with an independent heating system, and thus prevents the heat from being wasted by the neighbouring ovens.

(4) By this mode of construction the walls separating the ovens are necessarily massive, and form an efficient magazine for the retention of heat and the maintenance of the high temperature in the heating-channels when the neighbouring ovens are being emptied. The strength of these walls is such that the ovens can be constructed of a greater height, and consequently for the same capacity, narrower than is possible in any other existing system of coke-ovens.

(5) The heating-gas for each oven enters by three pipes, one into each side, one into the bottom-flue, and two into the upper horizontal flues, thus securing an advantageous distribution of the heat, and at the same time excluding the possibility of overheating of the walls.

(6) The air used for combustion is heated in flues and channels, situated below the ovens, before it is brought into contact with the gases. The gases, after having heated the ovens, can also be utilized to heat boilers, etc.

The method of heating the ovens may be described as follows:—

The bottom-flue is divided into three parts by cross-walls. The heated air for combustion (the temperature of the air at Kaiserstuhl was 420 degs. Cent.) enters the bottom-flue at each side of the oven. The gases from

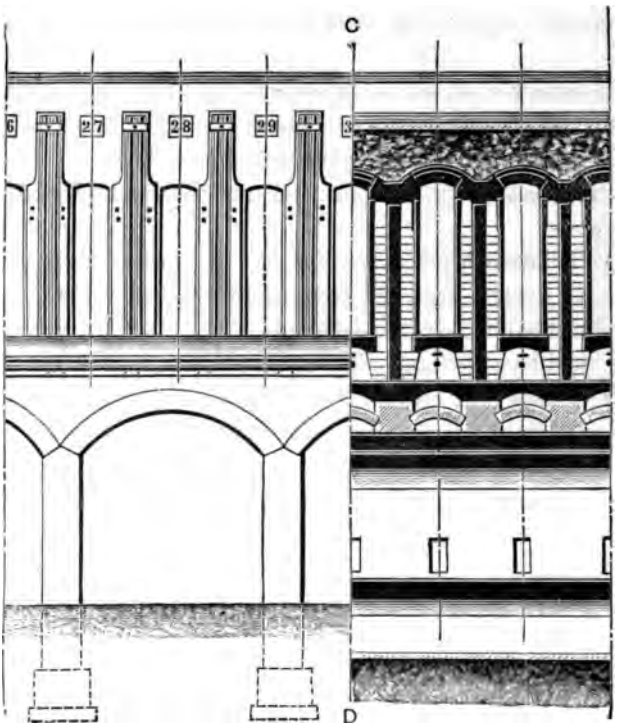


Fig. 1.—Front Elevation. Fig. 2.—Cross Section A B.

both sides are conducted through the middle series of flues downwards into the main flue below the ovens, along which they pass to the boilers.

The path traversed by the combustion-gases is a short one, and the gas used for heating is mixed with the air in the flues so intimately that the waste gases contain only $\frac{1}{2}$ per cent. of free oxygen.

The yield of bye-products obtained by this system of coke-oven is greater than with the Coppée-Otto ovens, which up to the present time,

have given better results than all other systems. This improvement may in part be attributed to the altered conditions of distillation in the oven,

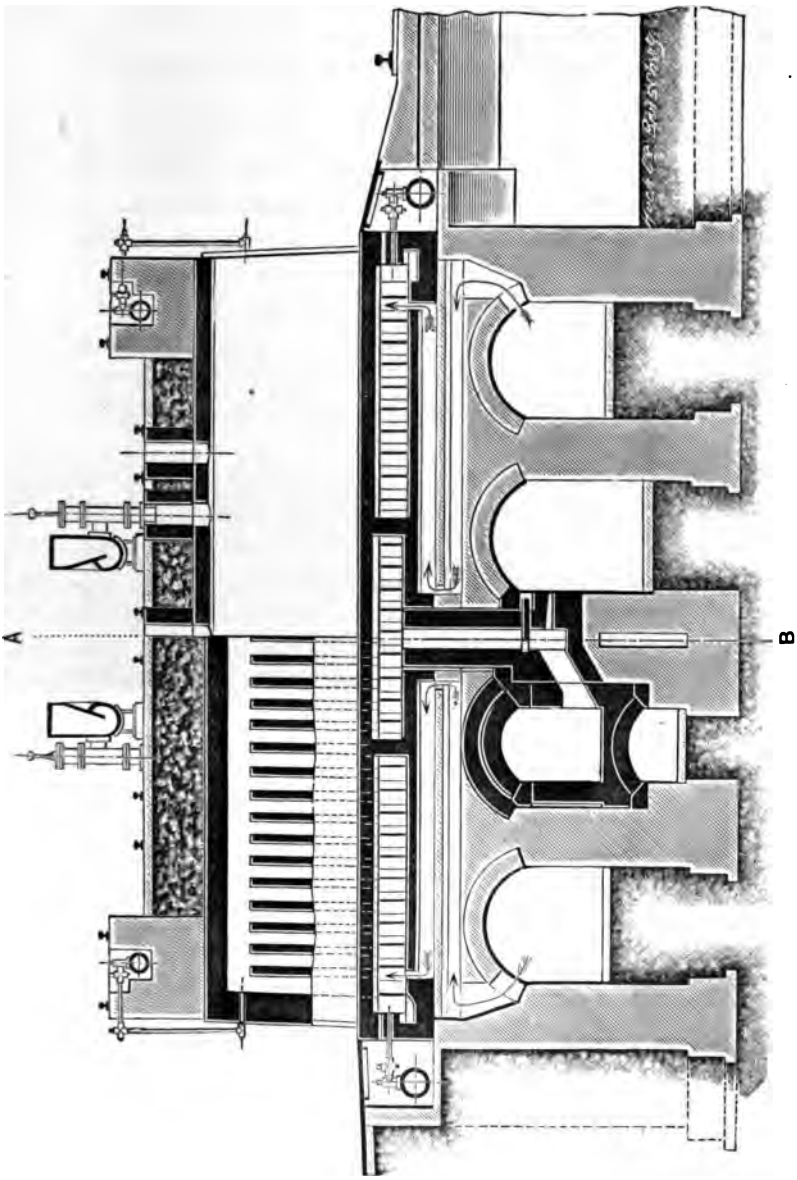


Fig. 3.—Cross Section C D.

but in the opinion of the inventor it is chiefly due to the fact that the obstacles to the flow of the gases in the flues are reduced to a minimum.

Consequently, there is little difference between the pressure of the gases in the flues and those in the oven; thus, equilibrium of pressure can be easily preserved, which almost entirely prevents the air used for combustion from entering the oven, and prevents at the same time the gas from escaping into the flues, although the joints in the several chambers cannot be kept perfectly tight. By this method of heating, no loss of bye-products can possibly occur, neither by the air used for combustion being forced into the ovens nor by gases escaping from them into the flues, without having previously passed through the condensing-plant.

The existence of obstacles in the flues interferes with the maintenance of this state of equilibrium of pressure, because they produce a difference of pressure between the gases entering and leaving the flues, whilst the pressure in the interior of the oven is practically the same throughout. These gaseous interchanges between oven and flues, and consequent loss of gases and bye-products will be the more considerable the greater the hindrance to the steady flow of gases along the flues.

Such conditions obtain in a marked degree in ovens with horizontal side-flues, as the heating gases must travel a considerable distance along narrow flues, the direction of which is repeatedly changed. In the opinion of the inventor this is the chief cause of the small proportion of bye-products yielded by ovens so constructed, as compared with the results obtained in the Coppée-Otto ovens. But even in this latter system the hindrances to the flow of the gases are greater than in the system described in this paper, because the whole volume of the heating gases is conducted into the flues from one side of the oven and has to be forced through one upper flue to the other side, whilst in the Brunck oven the gases are equally distributed in the two upper flues from each side, that is into four ways instead of one, and therefore the resistance to the flow of the gases is greatly reduced.

Favourable results have been attained as to quality and quantity of coke produced by the Brunck ovens. These advantages are to be attributed to the intensity of the heating process. The massive wall between two ovens retains a large amount of heat during each coking period, which is of service in the beginning of the next coking period when the flues are cooled by the introduction of a new charge of coal.

The independence of each oven is attained by the double series of side-flues, instead of one as in the Coppée-Otto and Simon-Carvès ovens. Thus, considering the process of heating of two neighbouring ovens in these systems, it will be noted that the temperature in the side-flue is lowest immediately after one of the ovens has been charged. It increases

gradually as the coking process proceeds, until the second oven is charged again. Now, the greater amount of heat is transmitted to the second charge at the cost of the first, which is stored up during the following coking period. When this first oven is again charged, it will do the same to the other one, taking away the greatest part of the heat for its own consumption. Consequently, in regular working, the temperature of the side-flues can never become higher than that attained in the middle of the coking period.

In ovens with a double series of side-flues the temperature rises continually up to the end of the coking period, without any reduction being produced by the loading of the neighbouring oven; the process is consequently accelerated, and a better quality and greater output of coke obtained. Owing to the equal distribution of the heat, the overheating of single parts is prevented, and the upper parts of the walls are heated

Fig. 4.—Plan of Bottom-flues.

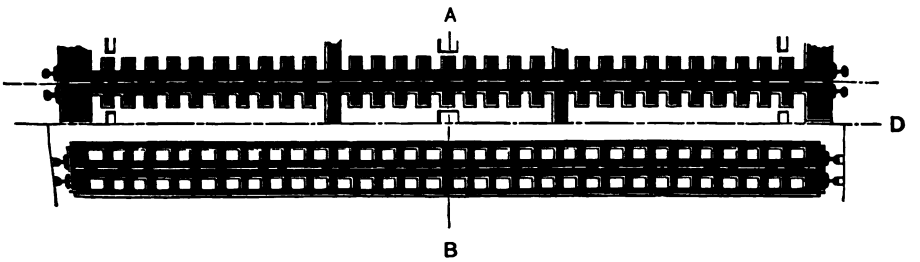


Fig. 5.—Plan of Side-flues.

intensely. This circumstance is of great importance, as the middle and upper part of the charge of coal is heated only from the two sides, whilst the lower part of the charge receives heat both from the walls and from the bottom of the oven. In the middle part of the vertical side-flues, the heating is generally imperfect, because the gases (partly used already) pass too rapidly through this portion of the flues. In the Brunck ovens this disadvantage is easily remedied. There being no great heating to fear, the thickness of the walls of an oven can be reduced as far as the quality of the fire-bricks used in construction will allow. The consequence is that the heat is transmitted more quickly to the charge of coal, and the temperature of the gases leaving the oven is lower. This temperature has been found at the Kaiserstuhl colliery to be 1,050 degs. Cent. in Brunck ovens, and with Coppée-Otto ovens working the same coal, 1,190 degs. Cent. Therefore the utilization of the heating gases in the Brunck ovens is the most perfect. Moreover, the massive and strong side-walls allow the ovens to be built higher, and in consequence,

for the same capacity, narrower than in any other system. And since the time required for coking increases with the width of the oven, which is greatly reduced in the Brunck oven, the time required for coking is as short as possible.

These points—equal distribution of the heat, favourable cross-section of the oven-chamber, and minimum thickness of the wall of the oven, combined with the independence of each oven, account for the better quality and greater output of coke.

Considering the fact that special arrangements for heating the air used for combustion are not required, and that by the central position of the main flue conducting the gases to the boilers the loss of heat must be greatly reduced, it is easily seen that in the Brunck oven the conditions for steam-production by the waste gases are exceedingly favourable.

The cost of erection and the working expenses are lower than is the case with other systems, owing to the ovens giving a better yield, and consequently the same weight of coke can be obtained from a smaller number of ovens.

The expenses for maintenance compare favourably with other ovens, because a partial destruction by overheating cannot occur, and the walls, owing to the lower temperature attained, seldom need repairs. Even when repairs are requisite, they can be performed without touching the massive middle walls and the vaulted roof, and without stopping any considerable number of ovens.

The practical results obtained at Kaiserstuhl colliery with the trial battery of six Brunck ovens are as follows :—The ovens are charged with 7·2 tons (7,200 kilogrammes) of coal, containing 14 per cent. of water, burning, on the average, for 34½ hours, the time for loading and discharging included; whilst in two large batteries of Coppée-Otto ovens, worked with the same charge of the same coal, the ovens must burn for 44½ hours. This rapid burning of the Brunck ovens is obtained, notwithstanding the small number of ovens, which renders it extremely difficult to maintain an equal heating and production of gases, as every interruption of each oven affects one-sixth of the whole production. Another disadvantageous circumstance is that due to the stoppage of all work at the ovens on Sundays from 6 a.m. to 6 p.m.

The total output of coke produced from the Kaiserstuhl colliery coal, containing 20·5 per cent. of volatile matter, is 1,250 tons per oven per year, whilst the Coppée-Otto ovens worked under similar conditions produce only 925 tons.

The yield of tar produced by the Brunck ovens is 13·4 per cent. greater, and of sulphate of ammonia, 9·5 per cent. greater than that of the Coppée-Otto ovens. These figures are obtained by comparing the best period of the sixty-two Otto ovens with the average yield of the Brunck ovens.

The steam-production by the waste gases, freed of benzol, and containing only ½ per cent. of oxygen and no carbonic oxide, reaches 820 gallons (3,650 kilogrammes) per oven per day; although the coals contain 14 per cent. of water.

The loss of heat in the small trial-battery is relatively much higher than would be the case in a larger one. Under normal circumstances the evaporation, by working the same wet coals, may be calculated at 990 gallons (4,500 kilogrammes) per oven per day. Deducting the amount of steam necessary for driving the plant for the recovery of tar and sulphate of ammonia, a surplus evaporation of 784 gallons (3,500 kilogrammes) per oven per day may be expected when coking wet coals of similar quality. When the coal contains a larger percentage of volatile matter, it has been proved by recent experiments that a constant amount of gaseous fuel, besides the waste gases, is available for heating boilers, etc. In this coking trial, with 300 tons of coal, containing 23·3 per cent. of volatile matter, 7·175 tons (7,175 kilogrammes) of coal containing 13 per cent. of water were coked in 30½ hours, representing an annual output of 1,375 tons of coke per oven.

The yield of bye-products has been so satisfactory that the inventor is able to guarantee for this description of coal a yield 15 per cent. higher than an estimate, based upon the results obtained by the Coppée-Otto ovens.

Mr. JOHN H. DARBY read the following paper on "The Semet-Solvay Coke-oven":—

THE SEMET-SOLVAY COKE-OVEN.

By JOHN H. DARBY.

In Northumberland and Durham, where large interests are connected with coal and coal-mining, and where naturally more care and attention has been paid to the development of these interests, it is strange that the question of better arrangements for the coking of coal should not have attracted more consideration.

With one or two exceptions, all the coke made in this district is carbonized in beehive coke-ovens. This statement applies almost equally to the whole of the United Kingdom. In South Wales alone has any extended attempt been made to coke coal in a retort-oven, and there no recovery of bye-products has been undertaken.

The reason for this state of affairs is possibly to be found in the fact that Durham possesses the most celebrated coking coal in the known world. This coal will coke, and coke well, in spite of the unfavourable conditions to which it is exposed in the beehive coke-oven. Then again the coal is cheaply raised, and the burning away of 10 to 20 per cent. of coal is not very noticeable. The beehive coke-oven is inexpensive to erect and maintain, and no great skill is required in conducting the coking operation. These reasons are borne out by the fact that in districts where the coals coke with difficulty great developments have taken place in retort-ovens, and such developments appear reasonable. For instance, in Belgium, the writer believes that no beehive coke-ovens are working at the present time, the whole of the coke used by the iron and steel-producing works in that country being made in retort-ovens. The large and well-appointed establishments of Messrs. John Cockerill & Co., Seraing, the Marcinelle & Couillet Co., Couillet, and the Phœnix Steel Co., Ruhrort, exclusively use Semet-Solvay coke in their high blast-furnaces, and recover the bye-products of the coking process.

It is not a fact that retort-ovens are only adapted for dry coals; it is possible to coke dry coals in retort-ovens, and make good metallurgical coke, while the same coal treated in a beehive coke-oven would either not coke at all or make very inferior coke.

Coals containing from 14 to 15 per cent. of volatile matter up to rich coals with 35 per cent. of volatile matter are coked with ease in the

Semet-Solvay oven, and the best possible coke from any class of coal under treatment can be obtained, while a net yield in residuals, at present prices, of 5s. per ton of coke made, after paying royalty, is obtained with fairly rich coal.

There are alterations to be made in the plant varying with the coal to be treated: as a general rule the Semet-Solvay oven should be narrow for lean coal so as to coke very rapidly, and wider as the coal is more highly charged with volatile matter. Consequently per ton coked, the cooling, exhausting, and washing-apparatus for the gas must be larger for rich coals giving large quantities of gas than for lean coals with a much smaller amount.

The following table shows results obtained from some coals used in Semet-Solvay ovens :—

Oven used.	District from which the Fuel was obtained.	Class of Fuel.	Volatile Matter, not including Water.	Yield of Dry Coke.	Percentage of Ash in Coke.
Semet-Solvay	Durham	Slack...	23·32	80·50	7·50
,,	South Yorkshire ...	,, ...	31·93	71·10	10·80
,,	South Wales ...	,, ...	17·20	81·82	7·10
,,	North Wales ...	,, ...	32·60	68·07	8·89

It would seem from the preceding table, that the percentage of coke obtained was greater than the coal could yield theoretically ; the weights were, however, carefully taken, and the author attributes the yield to the fact that large quantities of deposited carbon were found in the coke.

The writer has designed and erected a battery of Semet-Solvay ovens for the Brymbo Steel Company, Limited, North Wales, and when he was conducting enquiries as to the most suitable oven for making hard metallurgical coke for use in modern blast-furnaces, and one by which it was possible to recover the bye-products, he was struck by the large output of the Semet-Solvay oven compared with all the known systems now at work.

In Durham, there are retort-ovens which take about sixty hours to coke and produce 3¼ tons of coke per charge, the coal containing about 24 per cent. of volatile matter. The Semet-Solvay oven working on Durham coal, with the same percentage of volatile matter, produced 3 tons 6 cwts. of dry coke in twenty-two hours from one charge ; or taking the production of coke in the case of the retort-ovens referred to as 1·08 cwts. of coke per hour, the Semet-Solvay gave 3 cwts. of dry coke per hour. At many works, the Semet-Solvay oven produces 1,200 tons of coke per oven per annum, and this quantity has been considerable exceeded. In our district, it is 1,440 tons per annum.

The writer is informed, however, that faster working of retort-ovens has been obtained in Durham, and that charges are now coked in from forty-eight to fifty hours ; but this still leaves a great balance in favour of the Semet-Solvay oven with regard to rapidity.

Now, is this rapidly-carbonized coke as economical when used in the blast-furnace as other cokes? It is difficult to quote facts and figures from blast-furnaces in work, as the information so obtained is generally of a private character. The writer can, however, give two instances of blast-furnaces in work on Semet-Solvay coke (they are not in any way comparative, but simply show what can be done with the coke) :—

Furnace.	Ore used per Ton of Pig Iron. Cwts.	Limestone used per Ton of Pig Iron. Cwts.	Dry Coke used per Ton of Pig Iron. Cwts.
No. 1	36·40	7·30	16·70
No. 2	46·00	13·23	20·34

These results speak for themselves. In another case, the consumption of coke per ton of pig iron was reduced nearly 2 cwts. when using Semet-Solvay coke instead of beehive coke, other things being equal.

The writer attributes the great speed of coking, large yield of coke and bye-products, and ease with which repairs can be made :—(1) to the considerable mass of the supporting-walls ; (2) to the thin sides of the channel-bricks in the heating-flues on the side nearest to the oven ; (3) to the adoption of horizontal side-flues ; (4) to the arrangement of heating-flues on each side of one supporting-wall ; (5) to the fact that the coking process is carried on in closed retorts, so designed that leakage is reduced to a minimum ; and in this way the bye-products, ammonia, tar, light oils, etc., are not lost, but are carried forward with the escaping gas and fully recovered in the bye-product apparatus : the yield of coke is also at its maximum ; and (6) to the facility for changing all parts of the ovens subject to wear and tear, without disturbing the supporting-walls or main structure of the oven.

The Semet-Solvay oven very much resembles a large gas-retort, inasmuch as it is so constructed that it is a gas-tight chamber (Figs. 1, 2, and 3) lined on either side with large channel-bricks, containing the heating-flues, which are rebated one into the other so as to break the joints. The method of construction is very substantial ; thus at Havré, near Mons, in Belgium (where the first installation of Semet-Solvay ovens was erected), they have been at work for seven years without any important repairs having been made, and they are still in an excellent state of preservation.

By adopting this method of construction, and working the ovens so that a very slight pressure of gas is always kept on the oven side (as in

gas-works practice) the admission of air into the ovens is entirely prevented, no gas is lost by leakage into the heating-flues, and no coke is wasted by combustion with the oxygen of the air.

The following is an analysis of coke-oven gas working on washed fuel (after the absorption of the benzol vapours) :—

	Percentage by Volume.
Carbonic acid gas ...	3·27
Carbonic oxide gas...	7·95
Oxygen	0·23
Hydrogen	52·77
Nitrogen	1·99
Marsh gas	31·22
Olefines	2·57

The highly heated portions of the ovens, such as the channel-bricks containing the heating-flues, are independent of the main structure, and as they have only to carry their own weight, instead of the superincumbent brick-work, they are less liable to destruction.

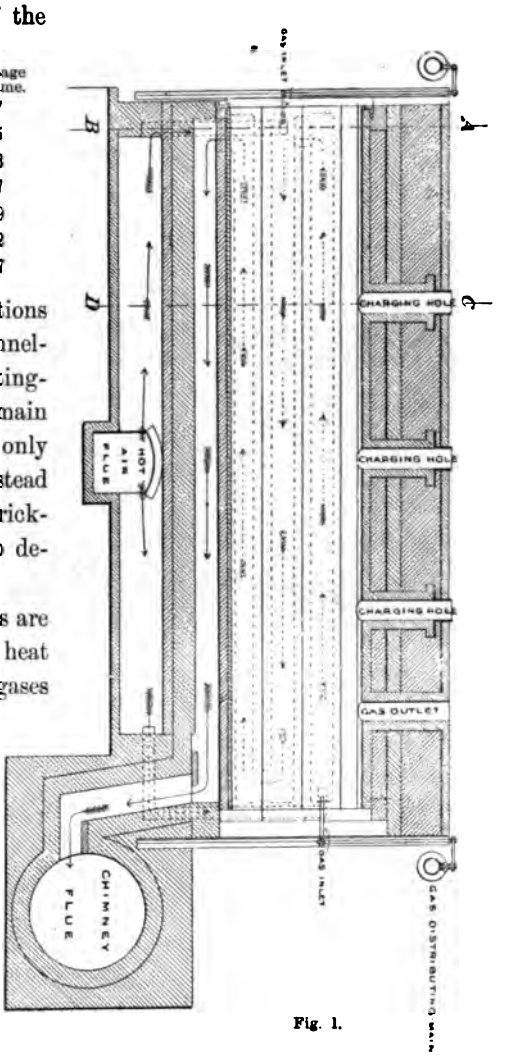
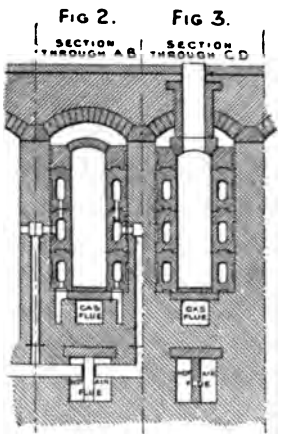


Fig. 1.

As the sides of the flues are thin, the transmission of heat is rapid from the burning gases to the fuel which is being coked. More efficiency is therefore obtained, and rapid coking without excessive heating also follows. The writer has found by experiment that when employing fire-clay walls to the side-flues, 2½ inches instead of 3 inches thick, 33 per cent. more heat was conducted through the thin than through the thick wall in a given time.

The supporting-walls between each oven also play an important part in regulating the heat. As soon as the coke is discharged from the oven,

the doors at both ends are closed, and the oven is filled with fuel as rapidly as possible. This necessarily cools the oven, but the supporting-walls, which are of considerable mass, give out their heat in the first instance to the channel-bricks, and maintain them at an even temperature. This recuperation of heat is important, or otherwise the oven would be apt to cool to some extent, and in working the ovens to obtain a good metallurgical coke, it is very important to submit the fuel to a high temperature as soon as it is charged, and that the coking process should be carried on without stoppage.



The Semet-Solvay ovens are usually built in batteries of twenty-five ovens, which makes a convenient size, and these twenty-five ovens should be discharged twenty-eight times in twenty-four hours. It is of great importance to the condensers, washers, etc., that charging should be continued, without intermission, day and night at stated intervals, otherwise the supply of gas given off is irregular, and interferes with the complete recovery of the bye-products.

The general dimensions of each oven are as follows:—Length, 30 feet; height, 5 feet 6 inches; the width varies according to the quality of the fuel to be treated, and must be determined for each particular case. For fuel containing only a small percentage of volatile matter, say 15 to 17 per cent., the width is only from 13 to 14 inches; for fuel rich in volatile matter, say 30 per cent., the width would be from 16 to 17 inches.

It is important to avoid loss of heat, in order to obtain equal distribution in all parts of the oven and to ensure the regular and rapid coking of the charge. With this object, a thick covering is placed over the chambers, in some cases as much as 4 feet thick. Sheet-iron doors are also provided to protect the cast-iron doors of the oven against the cooling influence of the weather. In this way a charge is coked right up to the fire-clay lining-bricks of the oven-doors, and no loss from half-burnt coke occurs at the ends of the ovens.

The doors are made so as to permit of easy and effective luting, and are raised from the top by means of a small hand-winch. All other openings, such as the charging-holes and the cleaning-holes in the ascension-

pipes, are closed by plates with faced joints, kept tight by springs in such a way so as to avoid the necessity of luting, and lessening the time occupied in charging.

The batches of coke, after leaving the oven, do not present any trace of wasting, which proves that the ovens are air-tight.

The gaseous products of distillation leave the ovens by the openings into the ascension-pipes and pass thence into the hydraulic-main, where the gases from the various ovens unite. The gas then bubbles through the water and liquor, from one side of the hydraulic main to the other, past the baffle-plate, and the temperature is quickly reduced, which prevents the deposit of hard tar in the pipes and recovery-apparatus. Each oven can be shut off by a valve from the hydraulic-main when being charged.

From the hydraulic-main, the gas, containing most of the tar, the ammonia, and the light oils, passes through the condensing apparatus. It is important, however, not to cool the gas too rapidly, or trouble will ensue by reason of deposits of naphthaline obstructing the pipes and recovery-apparatus and necessitating constant attention. If, however, the gas is slowly cooled, this drawback is entirely avoided.

During the cooling of the gas, portions of the tar and ammonia-liquor are condensed and run into the receiver, where they are separated by their difference of specific gravity. The tar is then pumped to the reservoir and the liquor to the ammonia-washer, as in the first instance it is too weak for treatment.

The gas, after leaving the condensers, is usually drawn away by a Gwynn or rotary exhauster, and forced towards the washers and absorbers, where the remaining ammonia and the light oils are extracted. The ammonia-liquor leaving the washer should contain 2·25 per cent. of ammonia, equal to 9 to 10 ounces liquor.

Before going any further, the writer would like to point out the benefits resulting from the use of fairly dry fuel in ovens, depending as it does on the great amount of heat rendered latent in evaporating water.

Take a battery, coking 100 tons of coal per day, and producing 19·7 cubic metres (696 cubic feet) of gas per minute. In the first instance the writer will give the number of calories to be absorbed per minute, in order to cool the gas which leaves the oven at 500 degs. Cent. down to 15 degs. Cent. on entering the exhauster, assuming that the coal is perfectly dry ; in the second place, the number of calories to be absorbed if the coal contains the normal quantity of water, viz., about 4 per cent.; and in the third case the number of calories to be absorbed if the coal contains 12 per cent. of water, which is a common percentage after washing.

Calories to be absorbed in each case per minute :—

(1) 19·7 cubic metres of gas	... × 0·34 × 485	= 3,248	calories.
(2) 2·77 kilos. of water per minute	× 814 = 2,254 + 3,248 =	5,502	,,
(3) 8·31 kilos. ,, ,,	× 814 = 6,762 + 3,248 =	10,010	,,

Consequently, if the above amount of work has to be done in cooling the gas and water-vapour, a similar amount of heat has to be expended in the ovens in raising the gas and water-vapour to the temperature named, with a corresponding consumption of fuel.

The gas, after it has been forced through the washer and the absorber, is conducted back in pipes to a distributing-main which runs round the ovens, and from this main it descends through regulators and service-pipes. The first supply to heat the ovens is burned with heated air in the top-flues on the back or ram-side of the ovens; passing along these flues the heated products of combustion reach the front of the ovens, where they turn down and meet the second supply of gas and air in the middle flue. The current is now towards the back of the ovens, where it again descends, passing forward to the front end, and finally the heated gases from either side unite in one main-flue under the bottom of the oven, by which they are conducted into the main or chimney-flue.

The division of the gases is intended to provide against the great local heating which would take place if it were all introduced at one point, and also to apportion the heat in a more uniform manner. The air necessary for the combustion of the gases is preferably heated in flues placed underneath the hearth. This air is drawn by the chimney, and has to travel through the heated flues before it is admitted at the point of combustion with the gases in the side-flues of the oven. The heating of the air is carried to 350 or 400 degs. Cent., and in this way a part of the heat which would otherwise be lost in the hearth of the oven is recovered, and the hearths are also kept comparatively cool.

Dampers regulate the communication between each oven and the main-flue for collecting the waste gases, and they are opened to such an extent that the draught of the chimney is equal in effect upon each oven. Through the main-flues the heated gases and products of combustion are conducted under boilers for raising steam, and afterwards pass away into the chimney. The Babcock and Wilcox is a convenient type of boiler for this purpose. It is a powerful boiler with a large heating-surface, occupies a small area, and offers little or no obstruction to the draught. The writer has had these boilers at work in connexion with coke-ovens of various descriptions for over eight years, and they have

given every satisfaction. The boilers should have a heating-surface of 70 to 80 square feet for each oven at work, and the waste heat is sufficient to evaporate 0·6 lb. of water per lb. of fuel coked. If the surplus gas is also burnt under boilers, 1·25 lbs. of water will be evaporated per pound of fuel coked.

The steam recovered from the waste heat leaving the ovens is amply sufficient for the requirements of the plant and the recovery of the bye-products.

In coking rich coal two-fifths to half of the gas produced is surplus, and can be applied for various purposes. An analysis of this gas has been previously given, and from a good gaseous coal 10,000 cubic feet per ton at least is obtained.

Before the absorption of the benzol, and consequent reduction of the olefines, the gas made from fair gas-coal is sufficiently good for illuminating purposes without enrichment, having an illuminating power of 16 to 17 candles.

The illuminating power of the gas is a very good indication of how the ovens are working. If it is high, no air is being drawn into the ovens, and the yield of benzol and other bye-products is large.

After the absorption of the light oils, the surplus gas, amounting to nearly 4,000 cubic feet per ton, can be used with great advantage in steel-furnaces, working gas-engines, or for generating steam.

The writer has calculated that if Semet-Solvay ovens were added to an open-hearth iron and steelworks plant, producing both pig-iron and steel, and if 2 tons 16 cwts. of coke could be manufactured per ton of steel made, no other fuel than the waste gases would be required for the manufacture of steel, including heating and rolling. As, however, this quantity of coke would probably make 2 tons 5 cwts. of pig-iron, and as 15 cwts. only are required for making a ton of steel, the rest being composed of scrap, it would either be necessary to make and sell 30 cwts. of pig for every ton of steel made, or sell about 1 ton 16 cwts. of coke. The coke-ovens would in this case be heated by producer gas, consuming about 2½ cwts. of small coal per ton of coke made, thus liberating the whole of the coke-oven gas.

The question of high cost has been urged against the Semet-Solvay ovens, but if the total cost of a battery be divided by the production of coke per annum, it will be found that the Semet-Solvay system is less expensive than any other recovery-plant per ton of coke produced, and the working expenses are not higher than those of the beehive coke-oven, although the cost of recovery of the bye-products is included.

The yield of bye-products from the Semet-Solvay ovens, with fairly rich coal, is about 5 per cent. of tar, 1·30 per cent. of sulphate of ammonia, and 3 gallons of light oil of a specific gravity of 0·89 per ton of coal.

The annual production of coke in the United Kingdom is probably 10,000,000 tons, representing at least 14,300,000 tons of coal. If this coal were treated for sulphate of ammonia, there would be an annual production of 181,000 tons from this source alone. So far, the demand for sulphate of ammonia has been in excess of the supply.

The exports of nitrate of soda from Chili amounted in 1894 to 1,082,000 tons, which at a value of £9 per ton makes £9,738,000. Whenever sulphate of ammonia approaches within £1 to 30s. per ton of the cost of nitrate of soda it replaces the latter for agricultural purposes, so that it will be readily seen that even if a large increase were made in the production of sulphate of ammonia, it would all be sold, and the country would retain nearly £2,000,000 annually which at present is spent in buying nitrate of soda from abroad.

The writer has personally examined coal from almost all parts of the world, and has visited all the important coal-fields. It strikes him as marvellous that with the best and richest coal in the known world, the inhabitants of the United Kingdom of Great Britain and Ireland should make so little use of the large quantity of valuable bye-products which it contains, while their foreign competitors collect and sell these bye-products.

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Mr. S. A. SADLER (Middlesbrough), wrote that he had had seventy Simon-Carvès coke-ovens working at his Lanchester colliery, near Durham, for more than a year, producing 1,000 tons per week of first-rate blast-furnace coke. The waste heat and gases from the ovens produced sufficient steam from four Lancashire boilers to drive all the engines connected with the bye-products plant and the colliery. Each coke-oven carbonized 5 tons 10 cwts. of coal every forty-eight hours, and produced an average yield of 76 per cent. of dry coke. The coal contained, on an average, 23 per cent. of volatile matter. By the aid of recuperating-flues the air required for the combustion of the gas under the coke-ovens was heated to a dull redness. Durham coal was evidently not so rich in nitrogen and volatile matter as the coal used in the Brunck coke-oven, and could not therefore give anything like the yields referred to in Dr. Brunck's paper. The bye-products that were being obtained from the Simon-Carvès coke-ovens at Lanchester approximated very nearly to those

claimed by Mr. Darby. They were as follow:—Sulphate of ammonia, 32 lbs.; tar, 7 gallons, and crude benzol, 2 gallons respectively, per ton of coke. Together these bye-products represent a value of 4s. to 5s. per ton of coke produced. It has been found that 18 cwts. of the Lanchester coke will produce 1 ton of pig-iron; it was therefore of essential importance that the ironmasters of the district should effect this saving, in addition to the increased yield of coke, which was at least 15 per cent. above that from beehive coke-ovens. Pig-iron manufacturers in this country could not meet the competition of German makers unless they adopted similar methods of making the coke in closed ovens, by saving the bye-products, and by the extra yield of coke, otherwise the foreigner would have an advantage of at least 6s. per ton of iron.

Mr. W. SCHROLLER (Durham), said that one of the chief characteristics of the Semet-Solvay coke-oven was the thinness of the sides of the heating flues, and they were separated by the bulky supporting-walls necessarily placed between them. In the coking of Durham coal (and it was the best coking coal in the world), it was difficult to believe that the Semet-Solvay coke-oven would give as good results as the Simon-Carvès coke-oven. The former coke-oven might certainly yield excellent results in coking dry coal, containing only 12, 14, or 15 per cent. of volatile matter, but different results would be obtained with Durham coal containing as much as 23, 24, and 25 per cent. Messrs. Simon & Carvès (now the Durham Coke & Bye-products Co.) had first introduced into this country and successfully worked bye-products coke-ovens whereby most excellent furnace coke was being produced regularly. The idea of thin coke-oven walls, he must admit, was very seductive, but nothing else; and as regards the quality of the coke produced by the Semet-Solvay, this is, with them, obviously a secondary consideration, because, as ammonia-soda manufacturers of world renown, all they care for in employing thin oven walls was: the production of ammonia. With the Durham coal owner, however, for example, the most material point was the production of good blast-furnace coke, and that has been proved by the Simon-Carvès construction by employment of thick flue-walls in daily practice and for years. He (Mr. Schroller) asked the author of the paper whether benzol can be recovered from the Semet-Solvay oven gases?

Mr. J. R. BRECKON (Sunderland) thought that it would be useful if the writers of the two papers would state the quantity of bye-products obtained; for instance, the quantity of tar in gallons, and the quantity of ammonia-liquor (8 degrees Twaddle) in gallons, so that the members

could see how the saving by the bye-products was calculated. A saving of 5s. per ton was very fascinating, and seemed to point to the necessity of investigation by competent authorities, in order to impress coke-manufacturers with its importance. The quantities of coke produced by these new coke-ovens, 1,375 and 1,200 tons respectively per oven per annum, were marvellous quantities, being four times greater than the yield of the beehive coke-oven, and considerably more than the Simon-Carvès coke-oven. Mr. Darby stated in his paper that a saving of 2 cwts. of coke per ton of pig-iron had been obtained. The use of coke made in retort-ovens in the Durham district pointed to the opposite result. The application of thin side-walls for the heating-flues appeared to be right in principle, provided that the durability of the oven was not sacrificed or the cost of maintenance considerably increased. He was glad that these papers had been brought forward, for the more the subject was forced upon the attention of users of coke, the more satisfactory it would be to the colliery owners, the makers of coke, the users of coke, and all concerned.

Mr. A. L. STEAVENSON (Durham) desired to disabuse Mr. Darby's mind of the fact "that the question of better arrangements for the coking of coal should not have attracted more consideration." He (Mr. Steavenson) read his first paper thirty-five years ago to the Institute on the manufacture of coke, and since then the question of the best coke-oven had received his continual consideration, and that of many others present at this meeting. He had tried many kinds of coke-ovens, but he still continued to work the old round beehive coke-oven. Mr. Darby stated that the Semet-Solvay coke-oven produced 80·5 per cent. of dry coke when burning Durham coal; but the best Durham coal did not contain so much. In an analysis made by Mr. John Pattinson he found that the fixed carbon was 68·74 per cent., and the volatile hydrocarbons 27·06 per cent. It was evident that Mr. Darby had produced unfinished coke, the coking process had not been completed, and (he supposed) the volatile hydrocarbons were remaining to a large extent in the coke. If the volatile hydrocarbons were left in the coke there was a considerable diminution of the work done by the coke in the blast-furnace. In all retort-ovens the coke must be withdrawn and watered afterwards, and the coke instead of containing only 0·4 or 0·5 per cent. of moisture contains as much as 4 or 5 per cent. Neither of the writers referred to the mechanical structure of the coke. At a colliery, in Durham, out of a possible 68·74 per cent. at least 64 per cent. of beautiful coke was produced, and in addition the waste gases heated all the boilers, pumped

all the water, and drew 900 tons of coal per day from a depth of 600 feet If they were able to produce first-class coke and do all that work in addition, they had every reason to be satisfied.

Mr. T. W. BENSON (Newcastle-upon-Tyne) presumed that Mr. Darby had been treating of coke entirely used at home for the special purposes required by the firm who made the coke. He would like to know whether the coke had been offered for sale in the open market, either for use by other people at home or for export. His experience of coke made from other than beehive coke-ovens was that it did not give satisfaction when sent abroad. About twenty-five years ago, in order to increase the produce of coke, his late father and himself applied bottom-flues and side-flues to circular beehive coke-ovens. The result was fairly satisfactory, as the ovens were burnt off in forty-eight as easily as in seventy-two hours, but the coke was formed in two distinct layers. There was a distinct fracture about mid-height in the coke, producing four instead of two "black-ends," and the coke was not so readily sold as the bright silvery product of ordinary beehive coke-ovens. In 1873, when coke was in great demand, he erected twenty rectangular coke-ovens, from which the coke was drawn in a body by means of an iron anchor and winch. The result was similar to the experience related by Mr. Steavenson, the coke was saturated with water, and had a tendency on being brought out of the oven to re-light and fire, causing an ash on the surface. The mechanical condition of the coke was not so good as that produced in the ordinary beehive coke-oven.

Mr. W. COCHRANE (Newcastle-upon-Tyne) said that twenty-five or thirty years ago he adopted a Belgian oven similar in type to the Semet-Solvay coke-oven, with a rammer for driving the coke out of the oven, and of the entire plant no remains were now left. The thin side-flues of the Semet-Solvay coke-oven were a source of weakness, and would speedily be destroyed with the removal of the coke. The coke would not be fit for blast-furnace purposes; its formation starting simultaneously from the bottom and sides, yielding a large quantity of breeze at the lines of separation and decreasing the percentage of valuable coke. The coke could not be cooled inside without considerable damage to the oven; therefore the coke must be withdrawn and watered outside, and would contain a large amount of moisture. All the various types of ovens, vertical and horizontal, had been adopted in the Durham district, but they had not found anything to equal the beehive coke-oven for the class of coal they used.

Mr. H. A. HUMPHREY said that he had seen the Semet-Solvay coke-ovens working on the Continent, in America, and had for the last few

years had a number of these coke-ovens under his charge. Messrs. Brunner, Mond, & Co., Limited, have at Winnington two batteries or a total of thirty Semet-Solvay coke-ovens. The primary object of these ovens was to make good coke and not, as Mr. Schroller had suggested, to recover the ammonia, although the latter was a most valuable product. At the present moment twenty-five ovens were at work, 109 tons of slack were coked per day, and thirty-one of these ovens were discharged per day. These ovens had been worked for weeks continuously at a coking-rate of less than nineteen hours per charge, a speed which compared favourably with any results of other coke-ovens. The extra temperature required for perfect coking at this speed of working was perhaps a drawback, and the rate first mentioned was to be preferred. The yield of coke was 80 per cent. on the slack, the coke containing 4 or 5 per cent. of moisture, an excess of water in quenching the coke when it leaves the ovens being used, as it is immediately filled into wooden waggons. He regretted that his experience did not extend to the use of Semet-Solvay coke for metallurgical purposes. At Winnington, the coke was used for burning limestone, and for this purpose a coke containing more ash was permissible. Washed slack, containing a large per cent. of moisture, had been used most successfully, but owing to the extra cost a cheap slack of the following analysis was now used:—

	Per Cent.
Moisture	2½
Volatile matter	23—27
Ash	12—16
Nitrogen	1·3

Owing to the high temperature of coking extremely little volatile matter was left in the coke. The quantity of gas produced had been underestimated by Mr. Darby. The quality of the gas produced at Winnington was shown by the following mean results of daily aspirator-samples taken during a month:—

	Per Centage by Volume.
Carbonic acid gas	3·0
Olefines ($C_n H_{2n}$)	3·1
Carbonic oxide gas (CO)	8·8
Hydrogen (H)	56·0
Marsh gas (CH_4)	23·7

or 91·6 per cent. of combustible gases; and with slightly less draught on the hydraulic main these results might still be further improved. The yield of ammonia was extremely regular and apparently depended little on the quality of the slack. The cooling plant was of insufficient capacity for summer work, but taking summer and winter together an average of 0·3

per cent. of ammonia on the slack was obtained; that was 1·2 per cent. of sulphate, equal to 27 lbs. per ton. The coke, so far from being soft as suggested by Mr. Cochrane, left the ovens in a long solid wall, and three men with rakes were required to pull it flat on the ground for quenching. By the use of a bubbling main there was no difficulty whatever with the formation of hard tar, and the cleaning of the plant was reduced to a minimum. The coolers and coke-scrubbers were not cleaned more frequently than once a year. Mr. Darby had made a passing reference in his paper to the use of coke-oven gas in gas-engines. He would like to point out the relative value of this gas as compared with good coal-gas and Dowson producer-gas:—

1 cubic foot of good coal-gas	yields	680	units of	heat.	
1 „ „ of coke-oven gas	„	510	„	„	
1 „ „ of Dowson gas	„	160	„	„	

The position which the coke-oven gas holds was clearly shown by these figures, and he might mention, to prove the quality of this gas, that it constituted 75 per cent. of the illuminating-gas used throughout the large works and the village at Winnington. The other portion of the surplus gas from the ovens was burnt under steam-boilers, as described in the paper.

Mr. B. Dodd (Durham) enquired whether the Semet-Solvay coke-ovens were drawn out at regular intervals night and day or in the day-shift only, as this would make a considerable difference as to the condensing-plant required? He would also like to know the height of the blast-furnaces in which the coke had been tried?

Mr. L. H. Armour (Gateshead) wrote that, after listening to Mr. Darby's paper on the Semet-Solvay coke-oven, he felt constrained to review one or two points raised by that gentleman. From personal contact with many coke-makers in various parts of the kingdom and inspection of their methods of working, he was prepared to say that they were not only anxious for improvement in their processes, but that they were ready to spend money to prove any matter likely to advance their interests. He thought that even the simple beehive coke-oven demanded, at most collieries, judgment and readiness of resource that were very far from the description "no great skill." The Durham coke-maker aimed at producing fuel suitable for the work of the modern blast-furnace. To this end it was essential that all the volatile matter should be expelled; and that this was secured by the beehive coke-oven there was no attempt to question. Burning from the top of the charge as the fire descended in a beehive coke-oven the gases from below are obliged to pass through the upper incandescent layer, and in passing they

are decomposed, the coke being thereby coated with hard carbon. This goes on until the charge is glowing all through, and this thermo-deposition of carbon at a high temperature gives the coke the much-desired silvery appearance. The writer, having conducted a number of experiments to test this thermo-deposition of carbon, it might be of interest to give some of the facts. Among the coal to be coked in a beehive oven were placed boxes containing articles of earthenware, say, 12 to 15 inches, below the top of the charge. These articles, of course, underwent the full heat of the charge, and met the rising currents of gases, and, when the charge was drawn, were found coated with carbon of the thickness of 0·02 inch and more. Strips of this deposited carbon subjected to a high electric current split into layers of gossamer fineness, being no doubt parted in the same way as they were deposited. The hottest part of a coke-oven is the charge, and the greater the heat the better will be the texture of the deposited coat. Let them consider for a moment the action of a gas retort. The most highly-heated part is the retort, and the hottest part of the retort is that on which the fire strikes, usually the bottom; hence the coat of carbon-scurf which forms on the inside by the thermo-deposition of carbon, and as the retort attracts to itself this carbon, the coke is left comparatively free from this deposit, therefore it is dark, as compared with coke made in a beehive oven. Mr. Darby has told us that "the Semet-Solvay oven very much resembles a large gas-retort," and its effect on the coke will also have some analogy. This statement is borne out by the darker appearance of the coke produced. Mr. Darby dwells upon the short time required to carbonize each charge in the Semet-Solvay oven, but this extreme hastening has another side, as the gases are driven off so rapidly and from all parts alike, that even were the coke in a state to take on a crust of hard carbon, there is very little opportunity for it, nor are the conditions otherwise favourable.

Dr. P. P. Bedson, in reply to a remark made by Mr. Sadler respecting the yield of tar and of ammonia obtained with the Brunck oven, pointed out that Dr. Brunck stated the ovens to yield 13·4 per cent. of tar more than the Coppée-Otto oven, and 9·5 per cent. of sulphate of ammonia more than the Coppée-Otto oven.

The President moved a vote of thanks to Mr. R. Brunck and Mr. J. H. Darby for their interesting papers, and the discussion was adjourned.

The following paper by Mr. B. H. Thwaite on "Engineering Heliography, or the Sun-print Copying of Engineering Drawings" was taken as read:—

ENGINEERING HELIOGRAPHY, OR THE SUN-PRINT COPYING OF ENGINEERING DRAWINGS.

By B. H. THWAITE, C.E., Fellow of the Royal Photographic Society.

Introduction.

Since the appearance of the author's contribution to the *Minutes of Proceedings* of the Institution of Civil Engineers, entitled, "Heliography, or the Actinic Copying of Engineering Drawings,"* published some nine years ago, this rapid, accurate, and scientific method of reproducing engineering and architectural delineations has become exceedingly popular, and there are very few first-class engineers' drawing offices without some form of heliographic equipment. When this first paper was read, the number of engineering ateliers in which heliographic methods were utilized was exceedingly limited.

Engineers readily acknowledge that this scientific expedient has been of great advantage to them, both on the score of cost of production, and on the more important one of absolute fidelity of reproduction. The characteristics of the heliographic process are so important that its rapid popularization is not at all surprising. Heliography, properly understood and practised, enables the reproduction of dimensioned, mechanical, geometrical, or freehand-drawing delineations to be made with absolute fidelity to the original. Heliography, under favourable sunlight or actinic conditions (actinicism meaning the chemically acting energy of direct or diffused sunlight), enables the most elaborate and detailed drawing to be reproduced in a small fraction of the time that would be absorbed in mechanically re-copying the original delineation.

Heliography permits the reproduction of an almost unlimited number of copies; it also enables by mere manipulation the production of a full-size elevation, plan, or section, from a half-drawn tracing, whenever such half-section is equal and alike in all particulars to the full section when drawn in duplicate. If the original or traced figure to be produced is figured partly on one side and partly on the other, or on the face of the tracing, this imperfection is not observable in the heliographic copy.

* Vol. lxxxvi., 1885, page 312.

Heliography possesses other advantages that may be appreciated from a commercial side, for instance the facility with which manufacturers and merchants can obtain copies of tracings lent to them for the preparation of estimates; this rapidity of the heliographic system has permitted manufacturers and merchants to tender or submit prices, at extremely short notice, who, probably, would not, but for this facile process, have had the opportunity of doing so.

Large drawings can also be built up of sheets reproduced by the heliographic process in a most satisfactory manner.

Drawing Office General Procedure.

The Preparation of the Original Tracing.—The following procedure is possible, as a consequence of the adoption of the heliographic copying method.

Instead of completing original drawings by inking them in, accurate tracings can be made, to serve as the office negatives; if alterations are made on the original drawings, new tracings can be made and marked and numbered, the original drawing being in pencil can be easily altered, whilst a comparison between the two tracings will immediately show the extent and character of the alterations, and the confusion often exhibited in an original drawing that has been altered, is thus avoided.

Besides, the new tracings would not show any obscurity or indefiniteness, and if marked consecutively and carefully dated, would show when the alteration was decided upon.

If the tracings are carefully preserved, they will form a record of the work as carried out. Each tracing should be accompanied by a heliograph (which should serve for use in the drawing office), in addition to the several duplicates required for the workshops. The tracing itself should be carefully kept, uncreased and unsoiled, for special reference, and for heliographic purposes.

The author does not intend to describe all the various heliographic formulæ, because a reference to his previous paper will enable the student to follow up the subject more exhaustively. The author will confine his remarks to a description of those processes that have won their way to popularity by the action of the law of the survival of the fittest.

Outline of Procedure.—The practical art of heliography or sun-drawing is simplicity itself. The procedure may be divided into three divisions. (*a*) The preparation of the original delineation or figure on translucent material, such as tracing-paper or tracing-cloth. (*b*) The application of a special and actinically-sensitive

paper to the face of the traced delineation to be reproduced. (*c*) The exposure of the delineation, with its accompanying actinically-sensitive paper to the effect of actinically active daylight, or in other words, to the effect more or less direct of solar light, or its near equivalent, such as an electric arc-light, of which the luminous rays have associated actinic or chemically active energy. It may here be stated that with electric arc-lights of 6,000 candle-power, both the Pellet (which requires half an hour) and the other cyanide papers can be actinically printed. (*d*) The removal of the sensitive paper, after adequate exposure to the influence of actinic or chemically active light, to the developing solution. (*e*) The application of the adequately exposed sensitive paper to the effect of the developing solution, for a sufficient time and in a proper manner, so as to leave on the surface of the sensitive paper, a distinct or well-defined, and more or less permanent, but absolutely accurate reproduction of the original delineation.

Outline Description of Heliographic Equipment.—To effect these five different processes peculiar to all methods of heliographic reproduction, an equipment of the following character is required:—(*a*) Tracing on translucent paper of a suitable character; (*b*) the actinically sensitive paper; (*c*) the printing-frame, by which the sensitive paper is exposed to the influence of actinic light; and (*d*) the developing-bath or baths by which the figure is permanently fixed.

Detailed Description of Heliographic Equipment and Agents.—A detailed description of the equipment as classified may now be given.

Tracing-papers.—The translucent character of the tracing-paper or cloth is of first importance. Strong yellow-tinted paper is not only unsuitable for rapid effect, but is inimical to accuracy of reproduction, because unless the rays of light falling on the plane of the glass of the printing-frame are directly at right angles to it, the thickness of the tracing paper tends to produce a distorted and indefinite image.

Special thin blueish-white, or almost white parchmented and alkaline papers, highly translucent, and only tearable by somewhat violent tension, are now manufactured, and are undoubtedly the ideal paper for heliographic use. The cost of such paper of thin alkaline quality, and for lengths of 66 feet by 30 inches wide, is 0·144d. per square foot, while a better quality costs 0·28d. per square foot.

After use, the tracings should, as far as possible, be preserved from sunlight, because by actinic and chemical action of light the paper is apt, not only to lose its transparency, but also to become brittle, and white, opaque creases and lines are the result of its being folded.

It is not advisable that the tracing should be folded, because the folds not only tend to destroy the translucency, but they make it more difficult to obtain a close contact between the tracing and the glazed surface (glass) of the printing-frame. The importance of a close contact although obvious, may be thus enunciated; wherever a tracing is in imperfect contact with the sensitized surface, the light-rays flow beneath the opaque lines of the drawing, and there is consequently no reproduction within the area of contact-imperfection.

If the tracings are required to be rolled, they should be wrapped around stiff cardboard or wooden rollers at least 1½ inches in diameter, and carefully stored in positions removed from actinically active light.

Translucent Drawings.—Ordinary drawing-paper can be made temporarily translucent by immersion in benzine, but in order to permit of its use for heliographic reproduction a rapidly sensitive character of paper must be employed. Special translucent parchment drawing-paper suitable for heliographic reproduction can now be obtained.

It is always advisable to cut the heliographic paper which is to be actinicized, of a size slightly larger than the tracing, so that its margin protrudes from under such tracing ; the effect of the actinic light on the exposed paper will be a measure of that under the tracing, being, of course, slightly less under the tracing, and in a degree depending upon the translucence of the latter.

The Printing-frame.—This part of the heliographic equipment permits of the sensitive paper being placed in contact with the figured tracing, under the influence of actinic light.

The degree of success in the process of heliography greatly depends upon the merits of the printing-frame. The apparatus should possess the following qualifications :—

It should permit the operator to place the figured tracing rapidly in position, and in close and equal contact, effected by well distributed pressure applied over the entire surface of the tracing. If the pressure by which the tracing and sensitized paper are brought together, is local in its effect, irregularities and false impressions are produced, besides which the wrinkles and creases produce shadows on the heliograph copy.

It is advisable to employ at least two sizes of printing-frames, one of which shall be large enough to permit the exposure of double elephant copies, 30 by 43 inches, the other and smaller are to be sufficient for taking royal copies, 19 by 26 inches.

When larger copies than the frame can take are required, a line in pencil should be drawn across the tracing so that the heliograph can be

copied in sections, which are afterwards fastened together with gum or any other suitable translucent medium, at the line of connexion. The author has obtained excellent and very large copies in this way.

The structural characteristics of the printing-frame itself are simple enough, and will be understood from the drawings of different types submitted, and from the actual examples exhibited.

The glass of good plate quality, free from blows, streaks, and all blemishes, either of inequality of thickness or of translucency, should be at least, for the double elephant size, $\frac{1}{4}$ inch in thickness, while for the smaller frame $\frac{3}{16}$ inch plate-glass may be employed.

In order to equalize the pressure of the backboard on the surface of the tracing and sensitized paper, thick, soft felt, at least $\frac{3}{8}$ inch thick, or a piece of trebly-folded flannel, will satisfy requirements. Indiarubber sheets, the full size of, and attached to the edges of the felt or flannel, will improve the equalizing action of the felt.

The edge-frame should be as strongly proportioned as convenience of portability will permit, and be rebated so as to carry the plate-glass and the considerable pressure brought to bear upon its surface.

The backboard of well-seasoned wood, if not in separate parts, should be stiffly constructed and well bound up. It should have two handles to facilitate handling. If the backboard is in separate battens, the centre batten should be the first to be applied—each batten to have its clamping-down springs. There are several methods of clamping-down the backboard to the felt. The pressure is principally applied by steel springs, either flat, helical, or coil.

In the Street printing-frames, an air-cushion is attached to the frame, and by mouth-blowing both an equable and a considerable pressure can be produced.

The author's method of mounting the printing-frame is shown in Figs. 1 and 2 (Plate III.). This arrangement enables the frame to be swung round at any angle, and to be reversed when placing the tracing and sensitized copy in the printing-frame.

The glazed frame is also available, and is not uncommonly used in the author's drawing office for tracing direct on to parchment or drawing-paper. A powerful light is placed beneath the inclined glazed surface, on the other side of which is the figured paper. The light passing through the glass and drawing enables the draughtsman to trace the figure on to the drawing or parchment-paper direct.

Methods of Exposing Printing-frame to the Influence of Actinically Active Light.—The printing-frame is mounted on special bearings, and is

easily moved about. The details of these bearings are shown in Figs. 3 and 4 (Plate III.).

When rapid effect is desired, and when circumstances admit, the Allott and Jones arrangements shown in Figs. 5 and 6 (Plate III.) can be adopted. Another arrangement for bringing the frame into proper exposure or so as to get the direct rays from the zenith, where only attic lights are available, is shown in Fig. 7 (Plate III.). The frame is hoisted up to the inclined glazed part of the roof or fan-light, counter-balancing arrangements being adopted to facilitate the raising of the frame to the desired position.

When at all convenient, the arrangements shown, or any others that permit of exposure in the open air, are undoubtedly the best. Glazed windows absorb much of the actinic energy of sunlight. The ideal position is to so incline the frame that it may face the sunlight at right angles, in order to obtain the direct rays from the zenith. When direct light cannot be obtained, recourse can be made to the Hayward or other forms of street-area reflecting-mirrors. Fig. 8 (Plate III.) shows the Thwaite printing-frame associated with reflecting-mirrors.

Of course it is not absolutely essential to expose the printing-frame in the open air. The author has obtained hundreds of copies with the printing-frame placed inside the office.

When electric arc-lights are employed, the printing-frames may be placed in a circle around the light, some 6 feet distant from it. The best arrangement for applying the arc electric-light for actinic printing is that adopted by Mr. W. H. Stanley, London, and is diagrammatically shown in Fig. 9 (Plate III.). Three printing-frames are arranged within a distance of 2 to 3 feet from an arc-lamp, and in order to obtain an equal light-diffusion, condenser-lenses are suspended from the casing of the arc-lamp. The current used has a pressure of 50 volts, and the quantity of energy is equivalent to 50 ampères. With an arc-light of this intensity satisfactory ferro-gallic (black and white) copies are obtained, with an exposure of about 3 hours. The necessary arc-light exposure when using ferro-prussiate papers is less, not more than 2½ hours being required to obtain satisfactory prints.

On Placing the Tracing in the Printing-frame.—The *modus operandi* is simply as follows:—The figured tracing to be copied is placed with its figured surface next to the glass of the printing-frame. The sensitized paper, with its actinically sensitive or prepared surface next to the tracing is carefully and uniformly pressed into contact with it. The felt is now laid carefully upon the paper, and in such a way as not to disturb the

position of either the tracing or the sensitive paper. The backboard is then placed on to the felt, the clamping-bars are fixed in place, and gradually screwed, or, by means of springs, pressed down, so that the two papers are uniformly pressed against each other, creases or crumples not being permitted.

No obstacle, such as a mullion or sash of a window, should be allowed to intervene between the (actinic) light and the printing-frame. The shadow of such an obstruction would distinctly mar the heliograph copy produced.

Cutting and Placing the Sensitized Paper.—With the ordinary ferro-prussiate process paper in which the figure is delineated in white-lines on a blue-ground no special anti-actinic precautions in cutting the sensitive paper and placing it into the frame need be observed. If the paper is rapidly removed from its case, and cut with its face on the glass of the printing-frame (sensitive face downwards) it will not be affected by the momentary exposure to the diffused light of the office.

With very sensitive papers it is advisable to remove them from their case and apply them to the printing-frame in non-actinic light. The author has for many years used ordinary yellow window-blinds with success; but if a permanent non-actinic effect is desired the windows should be glazed with ruby or amber-coloured glass, or be covered with yellow or amber-coloured "glacial" papers.

The Direct Procedure for Producing Different Heliographic Effects.

The various chemical formula for producing the different sensitizing surfaces are given in the Appendix. It may, however, be said that the different sensitized papers are produced so cheaply by manufacturers making a speciality of this class of paper that there is little, if any, economy in the individual preparation, and especially is this so with the Shawcross ferro-gallic paper. This paper can only be produced by special machinery, and as an example of the manner of approaching this subject by two industrial nations, it is a fact that this paper is being principally made in Germany. The inventor was not able to find a British manufacturer willing to provide the proper machinery, and abandoned his patent rights.

Outline of Various Processes in More or Less Regular Use.—The different processes may be taken in the following sequence:—(1) The production of a white-line reproduction on a dark prussian-blue ground by the ferro-prussiate process; (2) a dark prussian-blue line reproduc-

tion on a white-ground, the ferro-cyanide or Pellet process; (3) an ink-black line reproduction on a white-ground, the ferro-gallic or Shaw-cross process; (4) a black-line reproduction on a white-ground, the melagraphic, a private, process; and (5) a white-line reproduction on a black-ground, the platinotype process.

Suggested Specification for Sensitive Papers.—In ordering actinically-sensitive papers, the following are the necessary requirements:—(*a*) To be of a high degree of sensitiveness; (*b*) to have excellent keeping qualities under proper conditions of storage and climate; (*c*) guaranteed permanence of the heliographic reproduction; (*d*) to be workable in any weather; and (*e*) to be easy of manipulation.

Specimens of various heliographic processes are shown in Plate IV., Figs. 13, 14, 15, and 17 being printed by Messrs. Allott and Jones, of Liverpool, and Fig. 16 by Mr. J. R. Gotz, of London.

(1) *White-lines on Blue-ground (Ferro-prussiate) Process.*

Quality of Paper.—A satisfactory quality of paper should not, when folded, exhibit white creases at the folds. Ferro-prussiate prepared cloth must be preserved from the light, and even be cut in non-actinic light.

Cost.—Workshop paper, good and strong, 32 feet by 40 inches, costs 0·577d. per square foot. Extra thin, for foreign postage and for producing second and positive copies, 32 feet by 40 inches, costs 0·975d. per square foot. Ferro-prussiate calico, 32 feet by 30 inches, costs 1·278d. per square foot. Extra-rapid effect, or extra-sensitive paper is 30 per cent. more costly than the slower and less sensitive papers.

Exposure.—The length of exposure in the printing-frame depends, of course, upon the sensitiveness of the paper, the average times being :—

	For super-sensitive rapid papers.	For slow or only moderately sensitive papers.
In bright sunshine...	3 to 7 minutes.	15 to 20 minutes.
In diffused daylight,		
fair weather ...	5 to 10 „	20 to 30 „
dull weather ...	15 to 30 „	1½ hours to 2 hours.

With rapid or super-sensitive paper, special care should be taken in watching the exposure; the lines of the figure on the completion of the exposure should appear in a light tint, whereas with the slow-acting paper the lines should appear of a bluish-black colour.

Colour-changes.—The colour-changes under exposure to actinic light influences are :—

1st colour-period,	before	exposure,	yellow.
2nd „	after	„	greenish-yellow.
3rd „	„	„	greenish-grey.
4th „	„	„	bluish-grey.
5th „	complete exposure, olive-blue colour with metallic reflections.		

The more powerful the actinicism of the light, the darker the olive-blue colour resulting from the complete action.

Development.—The completely-exposed sensitized paper should be removed from the printing-frame and be vertically immersed in water in the Thwaite vertical-bath, in the manner shown in Fig. 10 (Plate III.), or if the horizontal type of bath be used, the water should be applied with a rose, allowing the effluent flow from the prints to be continuous. The sensitized surface that has been preserved from the actinic light is washed off, leaving the figure in pure white lines. The print is then removed to dry, being hung by the side of the vertical bath or from a cord-line. By the use of blotting-paper the action of drying may be accelerated.

By employing water heated to 90 degs. Fahr. in the bath the development is more rapid and more complete. Evidence of under-exposure is shown after washing by the paleness of the blue-ground, and of over-exposure by the imperfectly white and tinted blue-lines.

By a simple adaptation of the ferro-prussiate procedure blue-line figures on a white ground can be obtained.

Ferro-prussiate Blue-lines on White-ground Procedure.—A specially thin kind of sensitized paper is used to obtain a negative or white-lines on blue-ground copy. This thin paper is similar to that employed for foreign postal purposes, and can be obtained in lengths of 32 feet at the price of 1·33d. per square foot.

This negative heliograph copy is then, and instead of the tracing, placed in the printing-frame, and another piece of sensitized paper is placed with its sensitive surface in contact with the figured side of the negative, or in the contrary position to that adopted in the tracing procedure, in which the figured tracing is placed next to the surface of the glass of the printing-frame. The time of exposure is, under identical conditions of actinic influence, four times greater than is required in the first or negative procedure.

In washing the ferro-prussiate prints, where the Thwaite vertical-bath is not available, an ordinary indiarubber tube should be provided, so that the water may be delivered over the entire surface of the paper; if the pressure be strong enough the use of a brush will not be required. The improper use of a brush will destroy the integrity of the lines.

(2) *Blue-lines on White-ground (Ferro-cyanide or Pellet) Process.*

Paper.—Mounted or unmounted paper can be obtained.

Cost.—Calico-mounted paper in 32 feet lengths, 30 inches wide, costs 2·30d. per square foot. Stout paper in 32 feet lengths, 30 inches wide, costs 1·05d. per square foot, and 40 inches wide, costs 1·015d. per square foot.

Exposure.—In the highly actinic sunlight of springtime, 20 seconds; dull or indirect light, 2 to 3 minutes; winter light, 5 to 30 minutes.

The exposure should be checked as follows:—Take six slips of the sensitized paper, each slip is then to be placed behind tracing-paper slips, on which lines as delicate as those on the tracing have been marked, and on an identical character of tracing-paper. The slips are then placed in a small printing-frame with a backboard, having as many hinged partitions as there are slips, and exposed to the same conditions of light as in the large printing-frame. During the exposure, the slips can be examined to test the position of the colour-changes.

Development.—Pellet sensitized paper, on removal from the frame, may be kept for several days, if it is not convenient or desirable, on account of absence of actinic light, to effect immediate development.

The exposed paper should be transferred to a saturated solution of ferro-cyanide of potassium (3 ounces of yellow prussiate to each quart of water). This solution should be contained in a guttapercha-lined tray. The exposed paper should not be immersed in the developing-solution in such a way that the solution reaches the back, but floated with its sensitive surface in contact with the solution. This may be easily done by turning up the edges of the exposed paper, which we will in future refer to as the heliograph or print. The print should be exposed to the solution for 40 seconds.

After the completion of the development, the print should be floated face downwards upon clean water in an acid bath composed of 8 parts of hydrochloric acid and 3 parts of sulphuric acid to 100 parts of pure water, say 3·6 ounces of hydrochloric acid and 1·2 ounces of sulphuric acid to each quart of water; an alternate formula being 5 ounces of sulphuric acid to one gallon of water. The acids should be mixed together in an earthenware vessel, and be poured into a second tray and guttapercha-lined bath. From 6 to 8 minutes is adequate to allow for the acid reaction on the iron compounds, and for the removal of the redundant iron compounds by the acid solution.

If the fingers are stained in the developing process, the colour can be removed by washing them in a diluted alkaline solution of caustic soda

or potash. In order to prevent the staining of the hands or discolouration of the prints during manipulation, it is a good plan to wear white cotton gloves.

In order to ensure uniform contact with the solution a camel's hair brush should be carefully pressed over the back of the heliograph print, so as to expel the air-bubbles.

If the ground of the Pellet copies is not quite free from blue discolouration, it may be effectually bleached in the summer months by being exposed to a 4 per cent. solution of sulphuric acid; in the winter months the solution may be strengthened up to 6 per cent. of sulphuric acid. Properly exposed, perfect development has been obtained if the lines appear in dark blue on the original yellow ground.

Although Mr. Shawcross has a just claim to be considered the pioneer of the ferro-gallic process, which will be described further on, he strongly recommends the Pellet procedure and formulæ. He has made use of this process for many years, but he adopts a slight deviation in the developing process.

Pellet copies, properly washed and mounted, make admirable drawings, and this process was extensively used by the late Count de Lesseps in his engineering ateliers. They leave nothing to be desired on the score of permanency, as the examples of Lesseps' drawings shown by the author proved.

Special Method of Development.—Mr. Shawcross' modified method of developing will be understood by reference to Fig. 11 (Plate III.). A drawing or other plane board, *a*, of suitable size is inclined slightly from the vertical against the uprights *b*, which may be placed against a wall *c*. The exposed sensitized paper or print is pinned on to the board with its sensitive side on the upper surface. An enamelled metal and spouted vessel, *d*, is used for containing the solution of ferro-cyanide of potassium (yellow prussiate) in the proportions already described. The solution may be heated, and is poured over the surface of the paper in all parts, and the development takes place forthwith. The solution flows to the lower end of the board, where it is caught by the trough *e*, which is inclined towards an escape-pipe or tube *f*, into which it flows, and descends to the brown earthenware vessel *g*, provided with a cover *h*, so as to prevent the incursion of dirt or liquids. The vessel, *g*, should always be maintained full of a saturated solution of ferro-cyanide (yellow prussiate of potash) of which crystals are shown at *i*.

The advantages of this method of development may be summarized as follows:—(*a*) The developing process is visible; (*b*) the developing solu-

tion can be heated to a temperature appropriate for the season of the year; (*c*) less space is occupied; and (*d*) the cleanliness of the developer and the recovery of the used solution.

If the lines of the print during development have a tendency to spread, it is evident that the developing solution is a little too strong, or that the exposure has been inadequate. If the lines are imperfect in continuity, that is evidence of over-exposure.

Final Washing.—After the print has been exposed to the acid solution, it should be thoroughly washed, as described, for the ferro-prussiate process.

For making alterations in copies made by the Pellet process, a solution known as blue-solving is sold by manufacturers.

The acid bath may occasionally be cleansed with a potash dissolving solution.

(3) *Black-lines on White-ground (Ferro-gallic) Process.*

Since the author first introduced this process (invented by Mr. Shawcross) great improvements have been effected in its manufacture, and although the formulæ are given in the Appendix, nevertheless, except for research purposes, the author does not advise individual attempts to make this paper. Thanks to the intelligence of German manufacturers, satisfactory ferro-gallic paper can now be obtained from English stationers.

Paper.—It is a good precaution, the night before the paper is required to be used, to uncoil such part of the roll of paper as will be cut, as a better, denser, and stronger colour-tone will then result.

Cost.—Thin quality of paper in 32 feet lengths, 30 inches wide, costs 1·05d. per square foot; medium quality costs 1·05d. per square foot; thick quality costs 1·27d. per square foot; and mounted on calico, it costs 2·7d. per square foot.

Time of Exposure.—The colour-changes are: First period, before exposure, yellow; second period, after exposure, fainter yellow; third period, after exposure, still fainter yellow; fourth period, after exposure, white ground. The tint disappears, leaving the figure or delineation in the original yellow tint.

The exposed paper or prints may be stored in cases for subsequent development for as long a period as ten hours. If the exposure is not completed by the time the colour has set, it may be allowed to remain in the frame until the following morning, when the exposure can be extended to completion.

Development.—It is advisable, although not necessary, to submit the surface of the print to water-contact before exposing it to the developing influence, and this prior water-contact influence increases the density of the lines.

The developing-solution is composed of 40 grains of gallic acid per 1 pint of water. It should be stored in a guttapercha-lined bath. Another developing proportion is ¾ ounce of gallic acid (mixed in earthenware vessels) per 1 gallon of water. The solution can be employed several times. The solution will be serviceable for a longer period, if the development be deferred until some hours after the removal of the print from the printing-frame.

Immediately the print is placed in the solution (if tepid, so much the better) the yellow lines will become a very dark violet-black. It is also desirable to subject the printed surface after development in the gallic acid solution, to contact with clear water for five minutes, or until the print has no colouring influence on the water.

If the print after development is not quite white, but violet tinted, it is evidence of under-exposure, whereas if the lines are not black the print has been over-exposed. The prints can be hung to be air-dried. Ferro-gallic prints can be treated exactly as if they were drawings.

(4) *Black-lines on White-ground, Melagraphic Paper (the Marion Process).*

Paper.—This paper is probably based on the ferro-gallic process, but only requires pure-water development.

Various qualities of paper are produced, including a linen-mounted one.

Cost.—The paper, in 32 feet lengths, 30 inches wide, costs 1·50d. per square foot, and 40 inches wide, costs 1·52d. per square foot.

Exposure.—The exposure is similar to that described for the ferro-gallic procedure.

Development.—The face of the print should be carefully brought into close and complete contact with the developing-solution, which, for this paper, is pure water. The contact should be effected without disturbing the water, otherwise the lines of the figure will appear to be broken. The heliograph print should be left undisturbed in the water for half a minute. The print may then be brushed over with a soft camel-hair brush and hung up to dry. The water, after a minute's contact with the heliograph print, will become highly discoloured.

The print should be removed from this bath to another, in which a steady flow of water from an indiarubber tube is directed over the surface of the print.

Drying.—It is best to allow the print to dry, by being hung up and exposed to simple air-evaporation.

(5) *White-lines on a Black-ground, the Platinotype (Willis) Process.*

This process is only recommended for reproducing small drawings. It gives most exquisite and permanent reproductions, and it is also very sensitive.

Cost.—The cost of the paper, except for special application, precludes this beautiful process from being generally used. The price per 24 sheets, each 20 by 16 inches, may be reckoned at 1s. per square foot.

Exposure.—The entire process with this paper must be carried on in obscure and non-actinic light. A few minutes' exposure to ordinary daylight is adequate, and should be carefully watched.

The colour-changes are as follows:—First period, before exposure, yellow; second period, after exposure, pale greyish-brown; third period, after exposure, dull-orange hue. The last colour-change shows that the iron compounds have been reduced.

Platinotype prints need not be immediately developed, but they should be preserved from moisture by being placed in cases, in which there is a lid containing calcium chloride.

Development.—The surface of the print should be placed for at least four seconds in a solution composed of 130 grains of potassium oxalate to 1 ounce of water. The development will be instant and complete.

The print should then be washed for ten minutes in a solution composed of 10 parts of citric acid to 100 parts of water. Another, and perhaps better, solution is 1 part of hydrochloric acid to 60 parts of water.

The prints should be finally washed in copious relays of clean water for at least fifteen minutes.

Further Details of Procedure.

The Opacity of the Lines of the Delineation.—In the preparation of the tracing intended for heliographic reproduction, the first essential is the character of the lines of the figure, which should be of a high degree of opacity. The best liquid Indian ink should be employed, if prepared from the solid. Great care should be taken to make it sufficiently dense. Grey lines of imperfect opacity only give very imperfectly defined copies. The

opacity of the Indian ink can be increased by the addition of gamboge or chrome yellow, and the addition of raw sienna is highly recommended. A specially opaque Indian ink is sold by engineering stationers.

Shading.—Shading can be effected either by Indian-ink lines of varying breadth and distance, or by employing ink of variable opacity. The method employed in the author's drawing-office, and introduced by Mr. Vickers, is to shade with crayon and stump, by which most effective and pleasing results may be obtained. Lead-pencil shading may also be adopted, but not with such effective results as with the stump.

Colouring Negative Heliographs.—If variable coloured heliographs (white-figures on blue-ground) are desired, this can be effected, but with only partial success, and by employing a colour of high opacity for the tracing. This colour should be mixed with flake or Chinese white added in the necessary proportions to secure opacity. Instead of dead colouring, hatching in the different colours with the brush is very effective.

The coloured surfaces will be reproduced in more or less white tones on the heliograph, and on these surfaces the different colours, rendered opaque as described, can be applied.

Centre and Radial Lines.—Centre, radial, and dimension lines particularly for the ferro-prussiate process, are first drawn on the tracing with dotted Indian ink, or, if in full lines, with prussian blue, carmine, or raw sienna, to which is added a small portion of flake or Chinese white.

Characteristic Hatching.—A system employed on the Continent in use for negatives, or white-lines on blue-ground copies, as a substitute for structural materials classification, is the employment of characteristic hatching. The author has drawn up a suggested structural element classification code, shown in Fig. 12 (Plate III.). To avoid confusion, when employing characteristic hatching, a key or reference-table should be supplied.

Shrinkage of Heliograph and Tracing-paper.—In order to enable the exact shrinkage of important and complicated drawings, in which high delineative and scale-accuracy is important, it is advisable to draw on the tracing to be copied a scale in two directions at right angles to each other, and in exceptional instances in a diagonal direction as well. The shrinkage-scales will be reproduced in the heliograph copy, and will form an index of the shrinkage-action of the heliographic paper.

The shrinkage-degree depends, of course, on the texture of the paper, the denser and harder this is the less the shrinkage, while soft papers are subject to a much greater degree of shrinkage. The shrinkage of ferro-cyanide, ferro-prussiate, and Pellet papers is about 0·005.

Effecting Erasions and Additions to Cyanotype Heliographs.—For the ferro-prussiate copies, corrections and additions may be effected with a diluted solution of oxalic acid, or a solution of carbonate of potassium in the proportion of 1 part to 9 parts of water. Flake or Chinese-white colour may also be used.

For the Pellet process, a solution composed of 1 part of caustic potash to 28 parts of water is most effective for making alterations and erasures. The solution should be applied either with a quill pen or brush and be immediately blotted, to prevent the action from spreading. Oxalic acid may also be employed in the same proportion as the caustic potash.

A solution sold by heliographic stationers under the name of blue-solving enables additions to be made to heliographs produced by the Pellet process and in the same blue tint.

For the ferro-gallic process, a solution composed of 1 part of sulphuric acid to 99 parts of water, will enable erasures to be cleanly and neatly made.

Preparing the Heliographs for the Workshop.—When the ferro-cyanide or ferro-gallic heliographs are intended for the workshop they should be coated with hard white varnish. This will prevent the chemical character of the print from being damaged by stains from lubricating-oil, etc. For the rougher classes of work the prints should be mounted on linen, using fresh white paste (free from acid or alum) applied evenly to the back of the heliograph.

Excellent gummed paper on cotton for mounting heliographs, etc., can be obtained. The gummed surface should be damped with a sponge or soft brush, and the heliograph laid on and smoothed down from the centre with a soft cloth.

Excellent linen-mounted sensitized papers can now be obtained at such a price as to make their application more economic than the system of mounting ordinary-paper heliograph prints.

Tracing-Paper.—A good tracing-paper should possess the following qualifications. It should be extremely tough and be perfectly transparent. Alkaline papers are the best. It should not cockle or stretch with wash colour, nor deteriorate, either in translucency or toughness, with age. Tracing-cloth is an excellent medium for obtaining satisfactory heliographic prints.

APPENDIX A.—Sensitizing Solutions Applied to Paper.

(1) *Ferro-prussiate Process.*

The solution applied to the paper should be composed as follows :— Ferric ammonic-citrate, 140 grains; potassic ferri-cyanide, 120 grains; distilled water, 2 ounces. The solution should be kept in stoneware bottles, $9\frac{3}{8}$ ounces of ferric ammonic-citrate and $6\frac{1}{4}$ ounces of potassic ferri-cyanide being dissolved separately in pure water, and then made up to one quart of solution. The sensitizing solution should be applied to cream-laid paper, rolled and well sized by a flat damping-brush, 6 inches wide, or a tuft of cotton-waste, and the paper should be allowed to dry in the dark. The solution should be applied uniformly and just sufficiently to cover the paper. After drying in the dark the paper should be rolled and stored in special cases.

Theory of the Process.—The results are due to the influence of actinicism, as a reducing agent, on organic ferric salts, which are reduced to a ferrous state. Prussian blue is formed by the reaction of ferric compounds with potassic ferro-cyanide, while white ferrous compounds form a white soluble salt with the same liquid.

(2) *Ferro-cyanide or Pellet Process.*

The sensitizing solution is composed of the following constituents :— Sodium chloride, 3 parts; ferric chloride, 8 parts; hydrogen tartrate, 4 parts; water, 100 parts.

The solution is thickened by the addition of 25 parts of gum arabic. In order to prevent the solution from sinking into the paper it should be applied as rapidly as possible. The theory is the same as that of the ferro-prussiate process.

(3) *Ferro-gallic or Shawcross Process.*

The sensitizing solution consists of :—Gelatine, 150 parts; ferric sulphate, 60 parts; sodium chloride, 94 parts; hydrogen tartrate, 18·8 parts; ferric chloride, 150 parts; water, 110 parts.

The solution should be uniformly spread over the surface of unsized or specially treated paper, by means of a roller-pad, and the paper dried in the dark. It should then be dusted over with finely powdered gallic or tannic acid. The powder should then be thoroughly rubbed on to the paper, until it is brought into contact with every part of the sensitive surface, four coatings being applied. Some makers, in order to prevent

the sensitizing solution from approaching the size of the paper, pass the paper, before being sensitized, through very hot rollers, to harden the surface and prevent the sensitive solution from sinking into it.

Theory of the Process.—A ferric-gallate is formed by the combination of gallic acid with a ferric salt; this ferric salt, by the influence of actinicism, is reduced to the ferrous state.

(4) *Platinotype (Willis) Process.*

This process is patented, and the details of the sensitizing solutions are therefore not published. The theory of the process is based on the reducing action of a ferrous salt on metallic chlorides when exposed to actinic light.

APPENDIX B.—Specification of Heliographic Equipments.

(1) *White-lines on Blue-ground (Ferro-prussiate) Process.*

One printing-frame, either hung on standards on the Thwaite principle, or merely portable. The glass to be 41 inches by 28 inches, and ¼ inch plate, free from all blemishes. There should be a suitable size of felt pad.

The Thwaite vertical, or other ordinary horizontal or tray-bath should be lined with zinc and fitted with the necessary handles; and it should be of the dimensions suitable for developing the largest sized sheets that can be dealt with by the printing-frame. Cost about £5 10s.

(2) *Blue-lines on White-ground (Ferro-cyanide or Pellet) Process.*

Printing-frame and developing-bath as above.

One guttapercha-lined tray-bath of suitable dimensions.

One zinc-lined tray-bath, with handles, of suitable dimensions. Cost about £6 10s.

(3) *Black-lines on White-ground (Ferro-Gallic) Process.*

Printing-frame and developing-bath as above.

One lead-lined tray-bath and one zinc-lined tray-bath of suitable dimensions, and fitted with suitable handles. Cost about £8 10s.

The following gentlemen and firms, viz., Messrs. Bemrose & Sons, Limited, Derby; Eastman's Photographic Materials Co., Limited, London; Mr. J. R. Gotz, London; Mr. Richard Keene, Derby; Messrs. Marion & Co., London; Mr. Parry, C.E., Liverpool; Messrs. Smith & Co.,

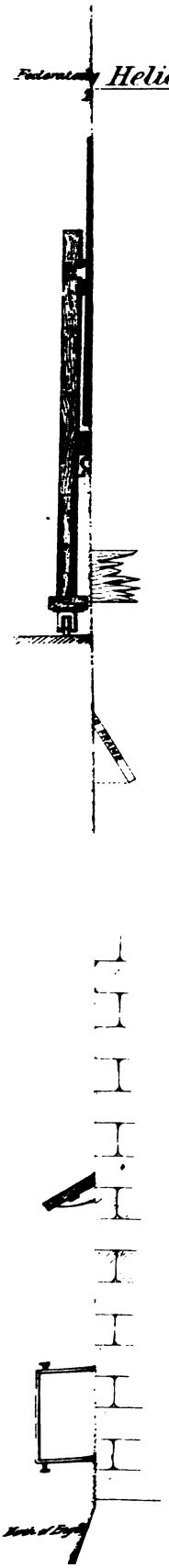
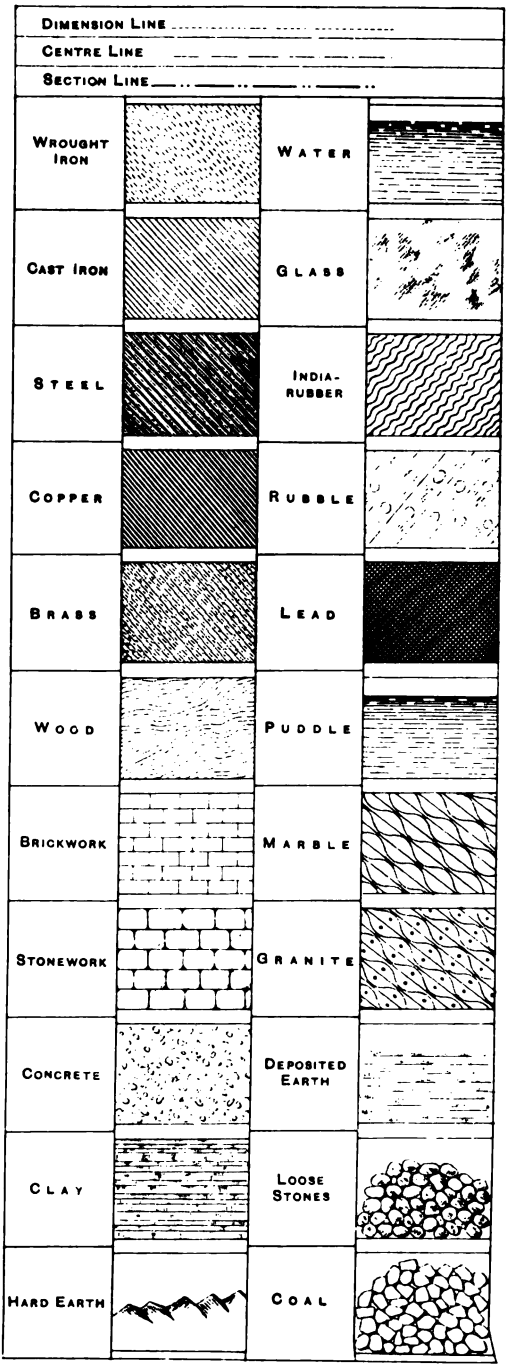


FIG. 12.—Characteristic Hatchings for Heliographic Purposes.



Manchester; Mr. W. F. Stanley, London; and Messrs. Thwaite, Tozer, & Co., Limited, London; exhibited examples of heliography by the ferro-cyanide, the ferro-prussiate, the ferro-gallic, the platinotype, and the baryte processes. A fine and interesting collection of every-day examples of heliography, as carried on under the supervision of Mr. Shawcross, C.E., of the waterworks engineer's department of the city of Liverpool, were exhibited. This collection lent by Mr. Parry, waterworks engineer, Liverpool, proved the permanency of ferro-cyanide and ferro-prussiate prints, a qualification further proved by drawings from the late Count de Lessep's ateliers, and by other French and Spanish examples nearly twenty years old. Some of the tracings and heliographs exhibited were prepared by Mr. Shawcross seven years ago; and both tracings and heliographs were in an excellent state of preservation. A comparison of the tracings with the heliographs, some of which showed patches, proved the readiness with which erasures could be made, and imperfections could be remedied. A very large heliograph cartoon (180 inches long by 30 inches wide) showing the dam of the Vyrny waterworks, illustrated how a very large scale drawing can be made by first preparing the details to one large scale (none of the drawings being larger than double-elephant or drawing-board size), then taking multiple copies and combining them up into a compact whole. The heliograph cartoon showed a long elevation to a large scale made up in this way in a most rapid and convenient manner, the only parts drawn in by hand being the varying rock-surface of the ground. The mounted heliographs shown appeared to be permanent and as effective as ordinary drawings. Other copies illustrated the original experiments made by Mr. Shawcross in converting the Pellet or ferro-cyanide (blue-line) process, into darker blue or black-lines. Other interesting exhibits showed the development of the ferro-gallic (black-lines on white-ground) process, of which Mr. Shawcross was the original inventor; the high state of perfection of this process was shown by numerous examples from well-known firms. Examples of a black-line on white-ground process, with merely a water-developer, were shown by several of the exhibtors, together with a series of platinotype prints by Mr. Richard Keene, of Derby. Examples of suitable tracing papers, printing frames, and other requirements of a heliographic equipment were exhibited. Eastman's Photographic Materials Co., Limited, exhibited examples of baryte paper. The methods of shading, colouring, and characteristic methods of colouring constructional drawings that have to be heliographed were shown by several exhibits prepared by Mr. Vickers of Messrs. Thwaite, Tozer, & Co., Limited.

This exhibition was the first of its kind, and should tend to the popularization of engineering heliography as an aid to engineering construction.

The PRESIDENT moved that a vote of thanks be given to Mr. Thwaite for his valuable paper.

The meeting was then closed.

NORTH STAFFORDSHIRE INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD AT THE NORTH STAFFORD HOTEL, STOKE-UPON-TRENT, MARCH 11TH, 1895.

MR. E. B. WAIN, PRESIDENT, IN THE CHAIR.

The SECRETARY (Mr. J. Richard Haines) read the minutes of the previous General Meeting, which were confirmed.

The following gentlemen, having been previously nominated, were elected:—

MEMBERS—

Mr. THOMAS ANDREWS, Colliery Manager, Talk-o'-th'-Hill Colliery, Stoke-upon-Trent.

Mr. WILLIAM JAMES CLARKE, Colliery Manager, Spring Hill, Madeley, Shropshire.

Mr. JAMES MADDOCK, Mining Engineer, Audley and Bunker's Hill Collieries, Stoke-upon-Trent.

Mr. F. R. OKE, Assoc. M.Inst. C.E., M.I.M.E., Crewe Works, Crewe.

Mr. ANTON RÄBER, Mechanical Engineer, c/o Messrs. Heenan & Froude, Birmingham.

Mr. JOHN WARD, F.G.S., Stafford Street, Longton.

Mr. W. N. ATKINSON said that the short paper he was about to read by Mr. Haarmann contained a "Description of the Arrangements at the Maybach Colliery (Germany) for Watering Coal-dust." Mr. Haarmann visited this country last year, and on his return to Germany he was good enough to send him (Mr. Atkinson) an account of the watering-installation at Maybach colliery. Thinking the account would be of interest to the members, he had asked Mr. Haarmann's permission to communicate it, this permission was willingly accorded, with the approval of the Royal Mining Office at Bonn. Maybach colliery has the large output of about 1,500 tons per day, drawn from the Albert and Marie pits, at depths of 1,260 and 1,500 feet, and a third level is being opened at a depth of 1,750 feet. Seven coal-seams, laying at an inclination of about 10 degrees, are being worked, varying in thickness from about $2\frac{1}{2}$ to $8\frac{1}{4}$ feet.

DESCRIPTION OF THE ARRANGEMENTS AT THE MAYBACH COLLIERY (GERMANY) FOR WATERING COAL-DUST.

By C. HAARMANN, Bergassessor, Friedrichsthal.

The system now in operation at the Maybach colliery for rendering the coal-dust harmless, is based on the regulations laid down by the Prussian Fire-damp Commission for dealing with the danger arising from coal-dust. The general arrangement consists of a network of pipes spreading throughout the workings, from which water under pressure can be ejected for thoroughly damping the dust at all such places where such precautionary measures are prescribed by the regulations of the Royal Mining Office at Bonn (see Appendix C), and at any other points where watering seems requisite.

The pipes are connected with a reservoir situated 492 feet (150 metres) above the middle-level, and 246 feet (75 metres) above the ventilation-level. This reservoir is formed in a heading 197 feet (60 metres) long and 10 feet (3 metres) by 8½ feet (2·60 metres) wide. The reservoir is closed off from the Albert shaft by means of a masonry-dam 7½ feet (2·30 metres) high, and its cubical content is about 11,000 cubic feet or 70,400 gallons (320 cubic metres). This quantity corresponds to about thrice the volume of water required daily.

The reservoir is supplied with pit-water by means of a pumping engine situated in the middle-level, and a pipe 3 inches (80 millimetres) in diameter. The engineman is advised when the reservoir is full of water by means of an overflow pipe from the top of the dam, connected with the engine-house.

The water flows out of the reservoir through a pipe 4 inches (100 millimetres) in diameter, connected with the bottom of the masonry-dam. The cleansing of the water entering this pipe from the particles of dirt contained in the pit-water, a point of considerable importance in view of the damaging effects of such dirt on the sprayers, etc., is accomplished in the following manner:—In the first place, the shape of the reservoir gives the dirt-particles in suspension an opportunity of sinking to the bottom; and, secondly, the water before entering the outflow-pipe is led through a filtering tank (Fig. 1, Plate V.), across which is stretched a wire-gauze of 0·02 inch (0·5 millimetre) mesh.

The outflow-pipe runs down the shaft to the hooking-place at the middle-level, where it is divided into two branches, each 3 inches (80

millimetres) in diameter, one of which passes along each of the two main levels or cross-cuts. In each of these branches a water-meter is placed so as to ascertain the volume used daily.

When the water is at rest the pressure in the pipes at the hooking-place on the middle-level is about 15 atmospheres, corresponding to the water-column of 492 feet. This pressure varies down to 5 atmospheres in the remainder of the system, according to the height at which given points of the pit-workings are situated relatively to the hooking-place. But if, in the regular course of working, water be drawn off, the pressure is subject to variations of several atmospheres. Even then, at the highest points of the workings the water spurts from the pipe in a spray several feet long.

The pipes branch off from the main cross-cuts through the haulage-ways, inclines, etc., to the face of the workings, and according to the amount of water required, the inner diameter of the pipes is diminished from 3 inches (80 millimetres) to $2\frac{1}{4}$ inches, $1\frac{1}{2}$ inches, and $\frac{3}{4}$ inch (60, 40, and 20 millimetres). The total length of the pipes amounts to 118,000 feet, or more than 22 miles (35,975 metres), of which

819	feet (	250	metres)	have a diameter of	4	inches	(100 millimetres).
1,623	„	495	„	„	$3\frac{1}{4}$	„	80 „
21,384	„	6,520	„	„	$2\frac{1}{4}$	„	60 „
23,421	„	7,140	„	„	$1\frac{1}{2}$	„	40 „
70,749	„	21,570	„	„	$\frac{3}{4}$	„	20 „

The wrought-iron pipes are used in lengths of $16\frac{1}{2}$ feet (5 metres); they are connected by flanges made tight with indiarubber rings, and are generally suspended from the roof by wires.

In accordance with the provisions of section 2 of the Mines Regulations (see Appendix C) the system of pipes is arranged as follows:—(*a*) In the main haulage-roads, self-acting inclines and travelling roads in connexion therewith, hose-stands are fixed at intervals of 164 feet (50 metres) communicating with the system of pipes by means of three-way pipes and stop-valves (Figs. 2, 5, and 6, Plate V.). (*b*) Similar arrangements are made in the communication ways between separate districts of the workings at certain points. (*c*) The same arrangement also applies to all working-places where coal is got; but here, of course, the three-way pipes are not used (Fig. 3, Plate V.). At the desire of the workmen, with a view to better ventilation and more rapid expulsion of explosive gas, stone-drifts are also connected with the pipe-system.

There are 545 hose-stands under classes *a* and *b* or roadways, and 232 under class *c* or working-places, with a corresponding number of three-

way pipes and stop-valves. In addition, there are 390 stop-valves in use on the branch pipes, so that when repairs or extensions are needed in any part of the system the working of the remainder need not be interfered with.

Indiarubber pipes are fixed to the hose-stands, the average length in the case of roadways, classes *a* and *b*, is 65½ feet (20 metres), and in working-places, class *c*, 5 feet (1·5 metres). See Figs. 2, 3, and 4 (Plate V.).

At the working-places, the watering is done by the workmen, but elsewhere it is done by special workmen, whose sole duty consists in keeping all the galleries constantly damp by spraying them.

When the water-distribution system was introduced, particular stress was laid on the installation of stationary sprayers in all the main roadways, in the belief that the air-current carrying along with it the minute particles of water would become saturated and thus keep all the roadways damp. But it was soon found that the air-current dropped the particles of water only a few feet beyond the sprayers, and consequently did not thoroughly accomplish the work of damping which had been assigned to it. Moreover, the sprayers were often displaced by the miners in order that they might not be wetted by the showers of water-drops while passing along the roadways.

For these reasons, the use of stationary sprayers has generally been abandoned: they are set up only where it is thought needful to increase the air-current on the injector principle at certain working-places, or where it is desirable to get rid of the accumulations of fire-damp which collect in hollow places in the roof. There are at present only twelve stationary sprayers at work. At the working-faces the use of the nozzles, which were formerly fixed to the end of the spraying-hose, has now been given up. It is true that the nozzles delivered the water in a fine spray, to the greatest advantage; but, considering the small cost of the water, the economy in that direction was not sufficient to warrant the employment of expensive nozzles, which often required repair or cleansing. The end of the hose is therefore made to terminate simply in a narrow pipe with a narrow opening, and sometimes even this pipe is dispensed with.

The whole system is placed in the charge of a special fireman, with five or six workmen under his control, and it is their duty to keep the whole of the appliances in working order. Their work consists in putting up and taking down the pipes in the main galleries, in fixing the hose-stands, in substituting new pipes and valves for damaged ones, in determining the daily consumption of water, and in controlling the working of the entire arrangement.

In the workings, the miners are required to fix the pipes and to remove them when they are working back.

The overmen of each district on their regular rounds have to make certain that the miners at the working-faces make a proper and sufficient use of the spraying apparatus. Moreover, the official who superintends the screening operations, takes note whether the coal, as it is emptied from the tubs, raises a cloud of coal-dust. If so, the men who are answerable for that coal are fined accordingly.

There are no other arrangements at the Maybach colliery for rendering the coal-dust innocuous. Nevertheless, with the view of further diminishing the risks arising from coal-dust and fire-damp, and in addition to obeying the Mines' Regulations, shot-firing in the coal is reduced to the lowest possible limit. In fact, blasting is only allowed where the toughness of the coal makes the winning by ordinary means a matter of too great difficulty. Gunpowder is prohibited, and safety-explosives only are used in such cases.

Among the accessory advantages which the water-spraying system brings with it are the facts: (*a*) that the miners do not seem to be so much disposed to have "sooty lungs" as formerly, and (*b*) that at the working-face they work in a fresher and cooler atmosphere. It is not possible to estimate numerically the effect which this may or may not have on the miners' coal-producing capacity.

Moreover, the water-pressure everywhere available in the pit under these conditions may be used for many purposes connected with the working. For instance: for the separate ventilation of special places on the injector principle, or by driving a turbine-ventilator; for working hydraulic engines for hauling out of dip places, or for drawing up *débris* required for stowing. The last-mentioned object is also attained by using the weight of the water as a counter-balance. Again, accumulations of water which may form above the deepest bottom-level are very easily got rid of by making use of the water-pressure to load syphon-pipes, or introduced as a jet to accelerate the running of long syphons.

Whether and in what manner the water of the spray-system can be advantageously made use of for working purposes in the mine, must depend on the conditions of each individual case.

Finally, it may perhaps be mentioned that the miners who on the conclusion of their shift travel home by train, make use of the spray-water to clear their hands and faces of dirt, there being no bath-houses above bank at this colliery.

APPENDIX A.—Cost of the Installation of the Water-spray System at the Maybach Colliery.

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6.	Spray-nozzles and ejector-pipes	71 ...	26	18	7
7.	Water-meters	2 ...	21	3	6
8.	Screws and packing-material	— ...	193	18	10
9.	Indiarubber hose	— ...	46	1	0
10.	Material for the reservoir, together with the filtering-tank	— ...	165	13	9
	Total		7,395	12	6
	Labour.				
11.	Wages paid for laying the pipes		410	1	0
12.	„ „ „ making the reservoir		249	14	0
	Total cost		8,055	7	6

APPENDIX B.—Cost of Maintenance of the Spray-water System During the Year 1893.

		£	s.	d.
1.	Wages paid for putting down and removing piping, replacing damaged portions of the apparatus, etc.	208	4	0
2.	Wages paid and materials used in the workshops for repairing damaged portions of the apparatus	17	18	11
3.	Wages of the superintending overman	79	2	4
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	Total	344	13	7
5.	Expenditure on watering at those points where this is not done by the workmen	96	9	0
6.	Cost of raising the water 492 feet (150 metres), from the middle-level to the reservoir, taking into account a *pro rata* of the use of steam power, of enginemen and firemen's wages, of cost of coal, and of repairs to the boiler, is found to amount to 0·0146 penny per cubic foot (0·044 mark per cubic metre). In the year 1893 1,213,740 cubic feet (34,368 cubic metres) of water were used, and therefore the calculated cost was 0·0146 penny × 1,213,740...	73	16	9
	Total expenditure	514	19	4

The production of coal during that period was in round figures 300,000 metric tons, and the cost per metric ton of coal (leaving out of account the redemption of capital cost of the plant) is therefore 0·412d.

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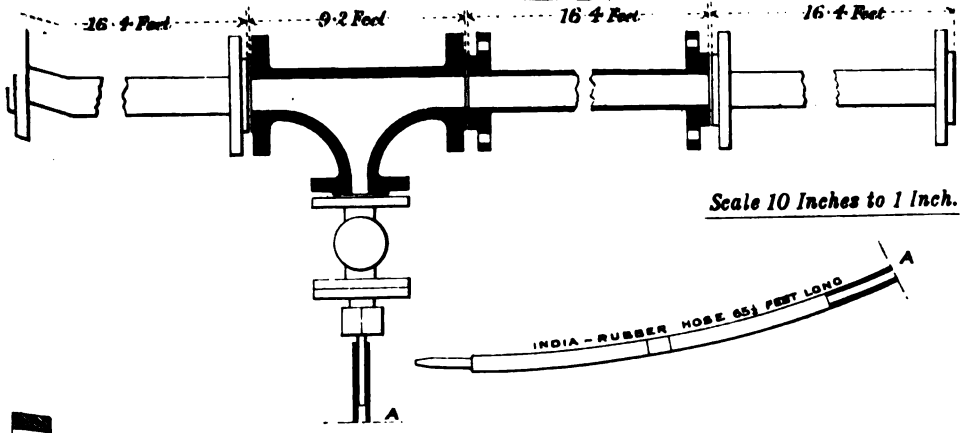
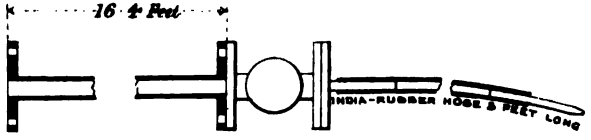


Scale 10 Inches to 1 Inch.

FIG. 3.



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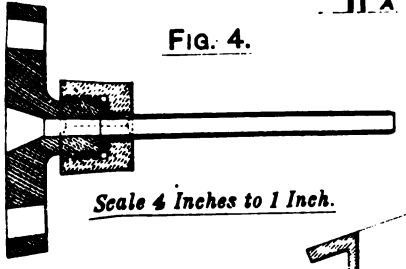

Scale 4 Inches to 1 Inch.

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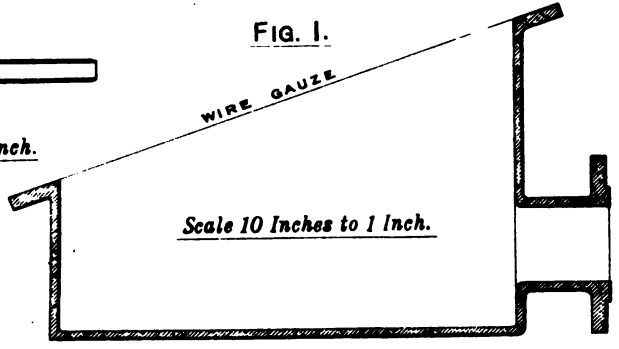

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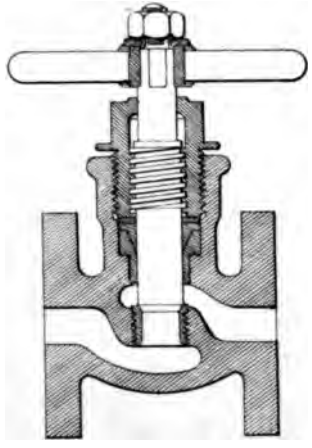
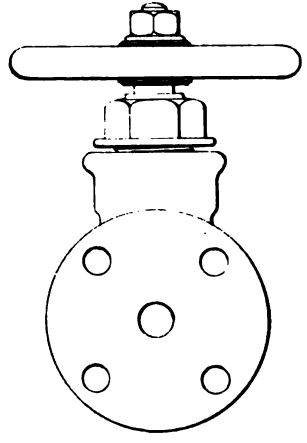


CROSS SECTION.

FIG. 6.



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APPENDIX A.—Cost of the Installation of the Water-spray System at the Maybach Colliery.

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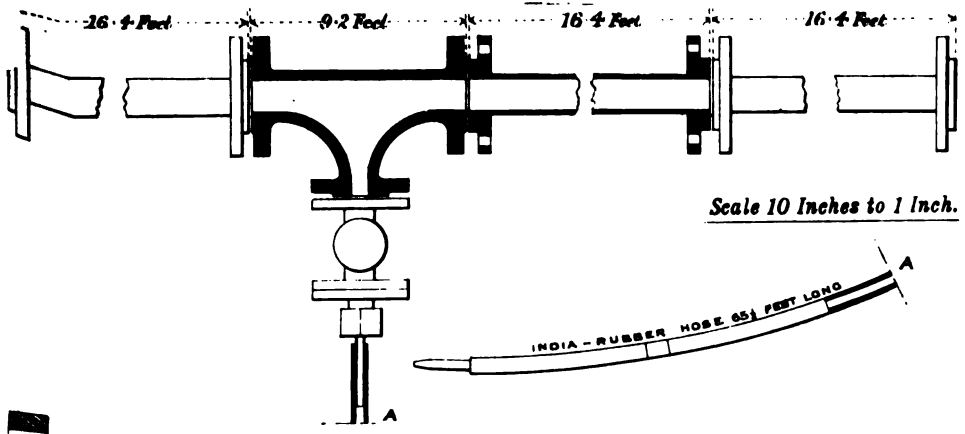

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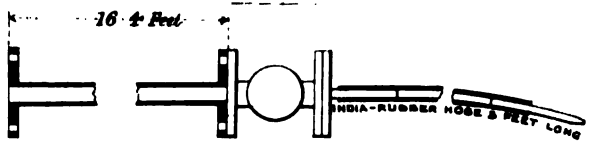

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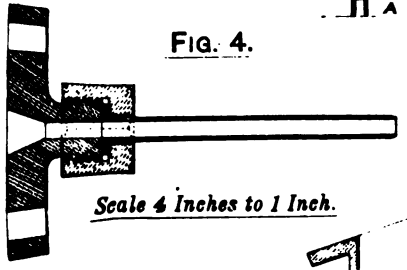

Scale 4 Inches to 1 Inch.

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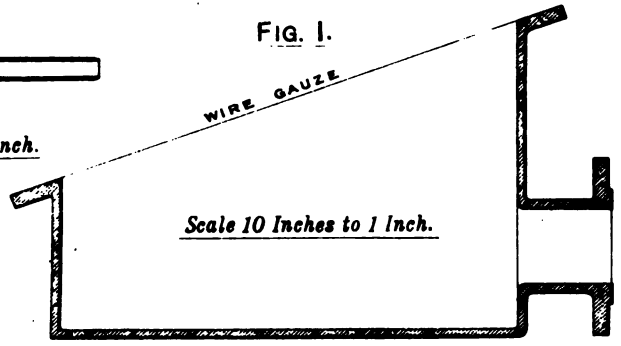

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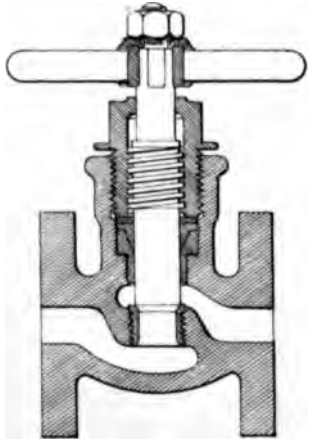
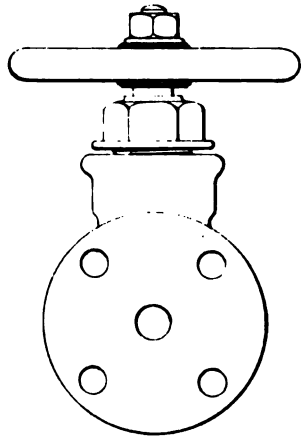


CROSS SECTION.

FIG. 6.



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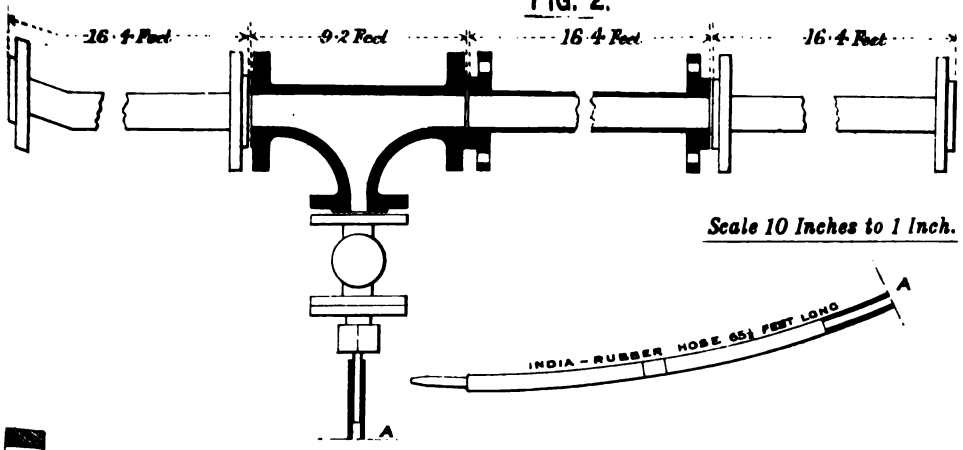

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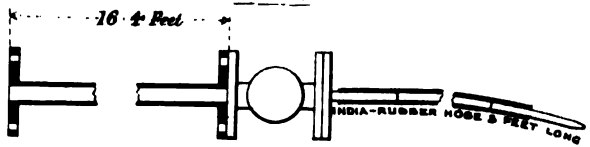

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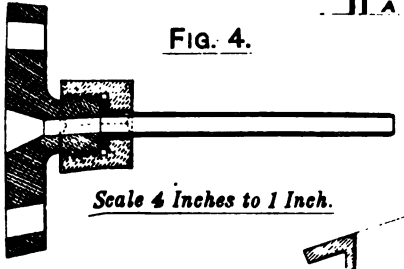

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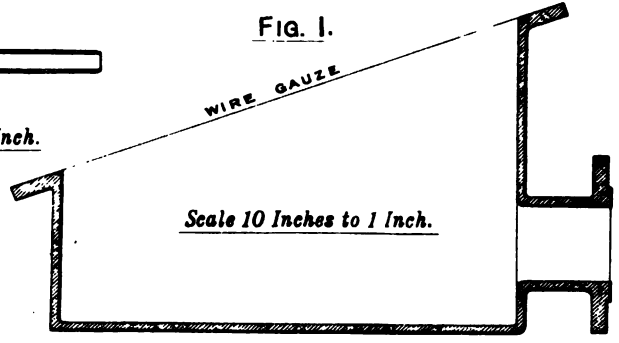

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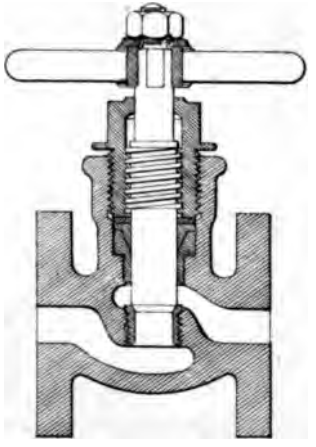

CROSS SECTION.

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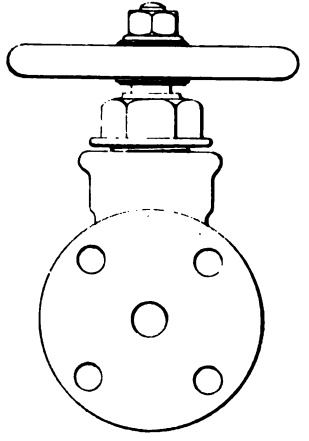

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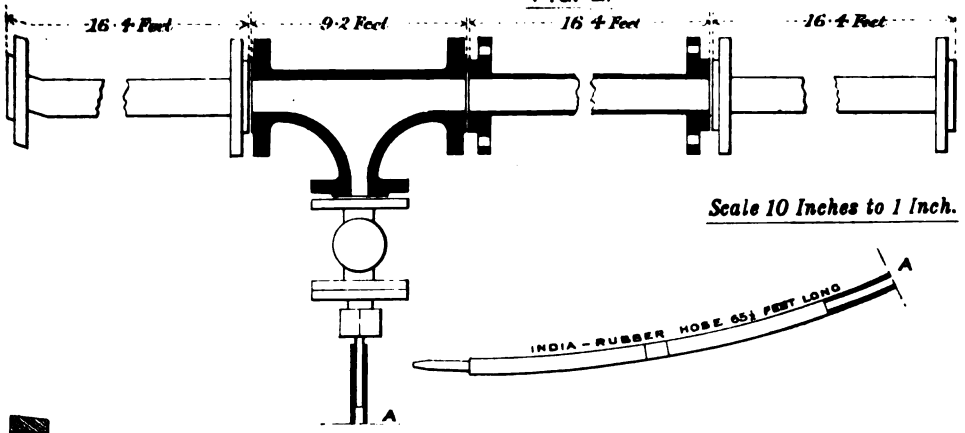


Scale 10 Inches to 1 Inch.

FIG. 3.

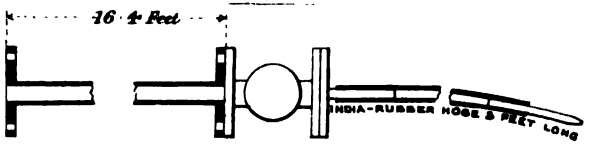
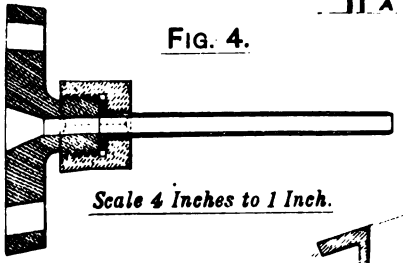
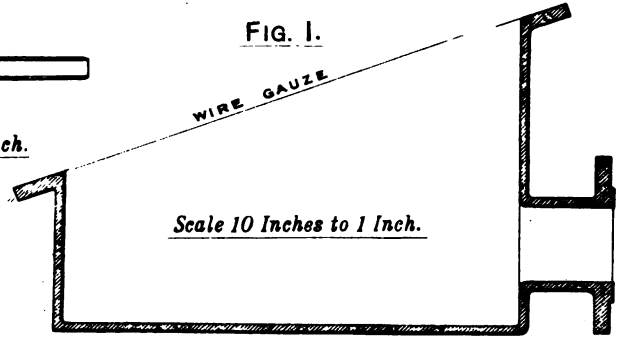


FIG. 4.



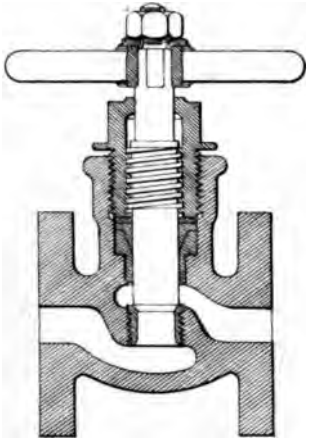
Scale 4 Inches to 1 Inch.

FIG. 1.



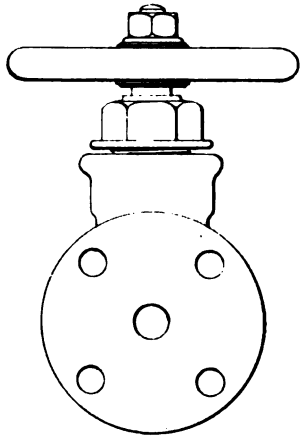
Scale 10 Inches to 1 Inch.

FIG. 5.



CROSS SECTION.

FIG. 6.



END ELEVATION.

Scale 4 Inches to 1 Inch.

APPENDIX C.—REGULATIONS FOR THE WORKING OF COLLIERIES CONTAINING DANGEROUS COAL-DUST IN THE SAARBRÜCKEN MINING DISTRICT, GERMANY.

Under sections 196 and 197 of the General Mining Law of June 24th, 1865, the following regulations shall apply to all collieries which contain dangerous coal-dust in the Saarbrücken district.

1.—Those collieries or portions thereof in which dangerous coal-dust is found shall comply with the following special rules :—

The Mining Office shall decide which collieries or portions thereof contain dangerous coal-dust.

2.—In those collieries or portions thereof referred to in rule 1, the coal-dust shall be thoroughly damped :

(1) In the principal drawing or haulage-roads and self-acting incline-planes as well as the travelling-roads in connexion therewith.

(2) In the other roads which connect different districts, so that the districts shall be separated from each other by sections of damp road of at least 164 feet (50 metres) in length.

(3) In all places in which coal is wrought.

The necessary arrangements for this damping shall be completed within a limit of time to be fixed by the Mining Office.

3.—Blasting with black-powder or slow explosives is prohibited. Blasting with other explosives may only be done by shot-firers, and then only when a radius of 33 feet (10 metres) from the shot-hole is thoroughly damped. The shot-firers must be paid day-wages, and be entirely, as regards payments, independent of the workmen placed under their charge. When the damping arrangements are not completed, or when they are out of order, blasting is absolutely prohibited unless explosives or methods of blasting are adopted by which the ignition of dust is rendered practically impossible.

4.—In exploring drifts, the duties of the shot-firer may be transferred to the leader of the shift, and rule 3, paragraph 3, shall not then apply. Blasting in stone-drifts is allowed without previously watering near the hole.

5.—Contraventions of these regulations are liable to be punished in accordance with sections 208 and 209 of the Mining Law of June 24th, 1865.

BRASSERT.

Royal Mining Office, Bonn, April 2nd, 1892.

Mr. J. J. PREST said he was strongly of opinion that the danger arising from the presence of coal-dust in mines would be most effectually overcome by the discovery and adoption of some absolutely flameless explosive. Mr. Atkinson had described a most efficient plant for watering the roadways of underground workings, although he should imagine that very few colliery owners could afford to spend £8,000 on such an installation, together with an additional charge of £700 or £800 a year for cost of maintenance and redemption of capital. Apart, however, from the question of cost, he had long held the opinion that the fixing of water-sprayers in a roadway, at greater or less intervals, to damp the dust would not prevent the extension of an explosion in such a roadway once it was set up, for the simple reason that the dust would not in such cases be thoroughly saturated with moisture; and it was only where a length of roadway in a particular district was absolutely wet, and not superficially damp only, that they might anticipate the watering of dust to be efficacious in preventing the extension of an explosion. This opinion was confirmed by the experience obtained at the Maybach colliery, where, out of a total of 777 stand-pipes, 12 only were fitted with permanent sprayers, the rest being utilized by the staff of workmen employed to attend to the plant, for fixing the hose-pipes and watering, he should presume, the different roadways in a manner similar to that in which we were accustomed to see members of a fire brigade use their fire-hose and nozzles. Then, again, assuming that a roadway or a length of roadway in each district had to be so thoroughly wet, so as to prevent the extension of an explosion or inflammation of dust, in how many of the dry and dusty pits of North Staffordshire could such a system of watering be adopted with impunity? Speaking from a limited experience, he had no hesitation in asserting that at any rate in the collieries under his management, where dust was plentiful, districts would be closed if he attempted to saturate perpetually any considerable length of roadway; he found enough difficulty already in keeping open roads in soft ground that were absolutely dry, and it would be madness in these cases to introduce enough water to lay the dust efficiently. Of course, it was possible that a length of, say, 50 or 100 feet of any roadway might be secured with a brickwork or steel-lining where the ground was otherwise difficult to keep open, and this length might be converted into a perpetual shower-bath by means of fixed sprayers, but the surplus water would probably have to be pumped out again from each district at a considerable cost, so that he failed to see how any system of watering could be employed with advantage universally to overcome the

danger, and as it was simply impossible to remove the dust: he could only hope that we might be put in possession of an absolutely flameless explosive as soon as possible.

Mr. G. A. MITCHESON said he was convinced of one thing—that some of the seams of the district would have to be left unworked if they had to be watered sufficiently to lay the dust, but whether that would apply to the greater number of them he could not say. He did not know how far the method of watering sections of the road would be effectual, so far as the spreading of the explosion by coal-dust was concerned. In one or two cases it would appear that where there had been short lengths of damp roads in the path of an explosion, the explosion had skipped the damp section and gone on to the dry and dusty portion beyond, so that damping in sections would seem not to be effectual. Of course, if they made the sections long enough, it would be effectual, but that would be nearly as bad as watering the whole of the roads.

Mr. JOEL SETTLE observed that the cost of the installation of the water-spray at Maybach was £8,000, and it was probable that if labour and everything had had to be provided in England it would have cost £20,000. There was £514 put down as the cost of maintenance, but nothing had been allowed for depreciation. They must allow 5 per cent. The best system had been described by Mr. Matthews, in a paper read before the Manchester Geological Society, in which it was suggested that water should be used in conjunction with compressed air.* Water could not be used in the seven-feet Banbury seam, which was largely worked in North Staffordshire. The floor was warrant earth, and if watered it became impossible to timber the places. There were other instances in which there was a peculiar roof which must not be watered. He never heard that the watering of a mine had prevented an explosion, although it might have checked its force. The fundamental principles to be observed were to keep all the working-places free from gas, to use the best kind of safety-lamp, to use the best high explosive, and to clean the roads from dust, watering them where practicable.

Mr. MITCHESON—How about gob-fires?

Mr. SETTLE said that if they watered the Bullhurst seam, it would produce a gob-fire.

Mr. B. WOODWORTH suggested that instead of 5 per cent. for depreciation, the allowance should be 12½ per cent.

Mr. J. RICHARD HAINES (Secretary) said, with reference to watering sections of a road, that about twenty years ago Mr. Macdonald described

* *Transactions of the Manchester Geological Society*, vol. xxiii., page 90.

an explosion at Harecastle colliery. The explosion had been curtailed by a wet cross-road, which prevented the extension of the effects of the explosion beyond; but of course it could not prevent the explosion. Mr. Makepeace had on a previous occasion given the Institute an account of the advantage of the use of an admixture of compressed air and water in South Wales, and no doubt he would again give them the benefit of his experience.*

Col. J. STRICK said that if they attempted to water in the seven-feet Banbury coal-seam the roads would lift, and in the eight-feet Banbury coal-seam the floor was almost like quicklime. The effect of using water, where they had longwall workings or old goaves below, would be that they would soon have gob-stinks, and in a short time gob-fires. He knew a case in the Bullhurst coal-seam, where they had never had a fire until a little fault was struck, water dripped through, and in a short time they nearly had a gob-fire. Wet roads would not prevent an explosion, but if the roadways could be kept watered the effect of the explosion would be reduced. He did not know how they could deal with the matter in the North Staffordshire mines.

The PRESIDENT said that the system described in the paper was as complete and comprehensive as anything that could be imagined. At one of their meetings, held four or five years ago, he suggested that it might, in extreme cases, be desirable to damp the coal at the face, but the idea was thought little of at the time.† He did not expect so soon to hear a paper telling them that such a thing had been done; but with all the elaborate watering arrangements at Maybach colliery there appeared to be still an objection to the use of black-powder, and though the danger from coal-dust must be thoroughly removed, the use of black-powder was forbidden by law. It looked as if in Germany the legislation had been grandmotherly, or that they were not satisfied with the efficiency of their own method. With regard to cost he estimated that 10 per cent. or 1·05d. per ton would be a fair allowance for interest and depreciation, but if they were to incur the expense of higher explosives it looked as if the first part of the expense was thrown away. It had been suggested in the discussion that coal-dust by itself would not cause an explosion; but he thought it had been quite conclusively proved that it had caused two explosions in the Radstock district of Somersetshire, where gas had never been found before nor since. As to the formation of dust, he did not think there was any real danger from it in the working-places, or in an ordinary jig road, a short distance from the face; but

* *Trans. North Staffs. Inst.*, vol. xi., page 20. † *Ibid.*, page 37.

there was a serious danger in main haulage-roads. The best way of preventing accumulations of coal-dust, as far as possible, was by having the tubs made so that they would not drop the coal on the floor. If the tub bottoms were made in such a manner that the coal could not fall through, there would be little complaint as to dust on the roads. If slow haulage and steel-bottomed tubs had been more largely used, they would have heard a good deal less about the coal-dust question than they had.

Mr. W. N. Atkinson said that no doubt the paper described a very complete system of watering, carried out to the extreme limits which it was possible to take such a system. They did not know anything as to the nature of the strata at Maybach colliery, and other matters which had an important bearing upon the application of water. A great deal had been said about the effects of watering on the strata. In that respect he believed that, owing to the nature of the strata, North Staffordshire was more liable to damage from watering than any coal-mining district in Great Britain. That was a very important point, but it was a question in his mind whether the difficulty might not be got over, to a great extent, by employing water in the form of spray. It seemed to him that although the damage might be very great if the water was applied in the form of a jet, if it could be used so as to simply damp the dust, it might be effectual without doing serious damage to the roads. Mr. Prest seemed to be of opinion that damping would be of no use, and that actual watering would be necessary. He could not say that he agreed with Mr. Prest, for he had seen the water-spray combined with compressed air in South Wales, and there it was effectual. There was a large quantity of dust in the road, sticking to all the surfaces, as dusty a road as they could find, and yet the dust was so damped that it could not fly about in the air, and therefore could not propagate an explosion. It would ball in the hand like snow, and in that state it was harmless. As to how far spraying would affect some of the strata in North Staffordshire the members had no knowledge, because it had not been tried on such a scale as to lead to any conclusion. As to watering of sections of roadway, that was a matter with regard to which we had no experimental data. He saw that in Germany they took it at 150 feet. For his own part he should be inclined to think that was rather short, but he had seen in more than one instance where an explosion was stopped, so far as he could ascertain, from no other cause than the lack of dust, and in three or four cases the distance was under 300 feet. He would not be prepared to say that 150 feet was not a sufficient distance, but he was quite sure that 300 feet would be. The cost,

of course, was an important matter, but if it was looked at from a broad point of view, this cost would not all be loss, that was, assuming the system prevented an explosion. If they considered the loss of property caused by an explosion, it would pay for a great many pipes, to say nothing of the saving of life. It had been stated that water had never been known to have prevented an explosion. That might be true; but he had never known that water had been adequately tried in a pit where there had been an explosion. There was room for improvement in North Staffordshire with reference to tubs, some of which would hardly hold the coal, let alone the dust.

Mr. MITCHESON moved a vote of thanks to Mr. Atkinson for having brought the paper forward.

Mr. J. J. PREST seconded the motion, which was agreed to.

The meeting then closed.

CHESTERFIELD AND MIDLAND COUNTIES INSTITUTION OF ENGINEERS.

GENERAL MEETING,

HELD IN THE TOWN HALL, LEICESTER, JANUARY 26TH, 1895.

MR. WILLIAM SPENCER, PRESIDENT, IN THE CHAIR.

The following gentlemen were elected :—

MEMBERS—

Mr. LANCELOT WALTER DE GRAVE, A.M.I.E.E., Electrical Engineer, All Saints' Works, Derby.

Mr. CYRIL EDWARD PARSONS, Metalliferous Mining Engineer, Birdsall Grange, York.

ASSOCIATES—

Mr. JOHN HENRY BAILEY, Deputy, Hall End Colliery, near Tamworth ; New Buildings, Dordon, Tamworth.

Mr. SAMUEL KIRK, Under Manager, Babbington Collieries ; Farwell Houses, Cinderhill, Nottingham.

Mr. ROBERT LOWES LISTER, Enginewright, Charity Collieries, Bedworth, Nuneaton.

The SECRETARY, by instruction and on behalf of the Council, gave notices as follows :—

1. That the termination of the financial year of this Institution, hitherto March 25th, shall in future coincide with that of The Federated Institution of Mining Engineers, viz., July 31st.

2. That the subscriptions falling due in advance on March 26th next shall, for the period then ensuing, viz., 1 year and 128 days up to July 31st, 1896, be 42s. 6d. for each Member and Associate Member, and 27s. for each Associate and Student (being at the same rate per annum as hitherto payable).

3. That a special general meeting shall be held in University College, Nottingham, on Saturday, March 16th next, at which the preceding notices will be submitted as resolutions recommended by the Council for discussion, and, if approved, for adoption.

4. That the meeting on March 16th shall be in place of the ordinary meeting in April, which will be omitted, and that at such meeting in March the usual notices in anticipation of the annual meeting must be given.

The PRESIDENT moved a vote of sympathy and condolence with the sufferers through the inundation of water at the Diglake colliery.

Mr. SMALLMAN seconded the resolution, which was carried *nem. con.*

DISCUSSION UPON MR. W. SPENCER'S PAPER ON "THE SUPPORT OF BUILDINGS."*

Mr. R. SMALLMAN (Nuneaton) remarked that the subject of the support of buildings was a very vexed question, and he did not think that it was possible for any one to lay down a hard-and-fast rule and say how many feet ought to be left. In working a mine, they approached buildings in various directions, and they found by experience that under different circumstances different proportions of coal had to be left. He would mention a few facts he knew of, respecting very inclined mines, as he apprehended that the members all understood the amount of support that ought to be left in flat measures. By inclined mines he meant mines with a dip of 1 in 3 to 1 in 5, and depths varrying from 200 to 1,200 feet. He had in his mind a case where they had opened out some workings at a distance of 300 feet from a school-house, surrounded by a wall, and the effect of the working, when the weight came on and caused the subsidence of the surface, was to break the wall at a distance of 345 feet; the school-house was to the deep of the workings, which were about 1,020 feet in depth from the surface. The thickness of the seam worked was 17½ feet of coal, less the waste. Endways on, the pull could not possibly have been so great, indeed it probably would not have exceeded 180 feet, about one-half of the pull from the deep. At another colliery, in similar measures at about the same depth, they worked within 60 feet of buildings on the basset of the workings, and there no ill effects were seen. In another case, they were opening a 6 feet mine lying at a depth of 600 feet, at an inclination of 1 in 3, and left about 300 feet on the basset of the shaft, which they thought was sufficient for its support, but it bulged over perceptibly, 9 inches or so. In another place, at a

* *Trans. Fed. Inst.*, vol. v., page 188; vol. vii., page 382.

depth of 510 feet, they worked from the deep and took the coal (6 feet) from under four houses, all close together, and never heard a word of complaint from the owners; the workings were started a longway to the deep side of the houses, and they got the weight on before they worked under them. The premises went down 3 or 4 feet, but the surrounding land did the same, and as they went down quietly the subsidence was not noticed. Speaking generally, if mines were tolerably flat one could work within one-third of the depth, according to the old rule, without risk. But even then one must be governed by circumstances. In another place, in Staffordshire, where the thick coal-seam(30 feet) was worked at a depth of 900 feet, mining operations affected some houses lying to the basset at a distance of 750 feet from the workings.

Mr. SOUTHERN enquired, when a shaft-pillar was mentioned, whether the radius or the diameter of the circle was given?

Mr. SMALLMAN replied that the radius was meant. It was a very old rule, and one that could be used without thought. If there was rock near the surface, and working had commenced, and the first break happened near buildings, the pull was worse than if there was soft ground under them. The pull would extend with soft ground and go gradually, and not suddenly, as it did in the case of rocky strata. He did not think that it was possible to lay down any hard-and-fast line as to the distance that must be left. One had to take all circumstances into account, the depth, the inclination, the nature of the strata, whether there had been previous workings in the upper measures, etc. It could, however, be generally assumed that in steep mines they had to leave a greater distance on the basset side than on the deep side of any buildings.

Mr. BINNS said that was known as "Callon's law." It was described in Prof. Callon's *Lectures on Mining* (translated by Mr. W. Galloway and Dr. Foster), and illustrated by beautiful diagrams.

Mr. SMALLMAN said that it was better to take out all the coal, than to leave insufficient pillars. The amount of coal to be left depended entirely on the strata, on the inclination, and on the position of the houses with respect to the first weight. If the weight came on close to the buildings, there was great danger, and the damage was always very much worse on the line of the first break. He recollected working in some measures about 5 feet thick, at a depth of 210 feet, with a deposit of drift above them, and they affected a railway-bridge at a distance of 450 feet. Of course this was an extreme case, but if they worked under drift with water, damage might be produced at any distance.

Mr. J. A. LONGDEN (Stanton-by-Dale) said that he had jotted down a few facts with reference to the subject, which was a very important one, and he was really surprised that it had not been taken up more fully. If mining engineers expressed a definite opinion as to the dimensions of pillars to be left at certain depths, probably some day a solicitor would read the *Transactions* and ask them in cross-examination, if such was their opinion, and it might be a little awkward and involve one in unlooked-for consequences. As regards the subject of the paper he thought that the most important consideration was the support of the shaft, for after all the shaft was the most important building about a colliery, and if they could not keep the shaft uninjured, and were thus unable to get their coals out, the colliery was not of much use. He had come across several cases of creep in shafts. He knew of a lease where it was stated that the shaft-pillars were to be left 300 feet wide. There were five seams leased in that particular royalty, and in was evident that the difference in the distance from the top seam at 450 feet, to the bottom seam at 1,350 feet showed that the fixed dimensions of the pillars was a mistake. In leases, in his opinion, it ought to be inserted that the shaft-pillars should be as wide and as long as the seam was deep, as a general rule. At Pleasley colliery, at a depth of 1,560 feet, the shaft-pillar was left 2,400 feet long by 780 feet wide. The pit-bottom did not run in, but it was a source of great anxiety, and had been rebricked three times, because the shaft-pillar was of the wrong shape. Although it had an area of 1,872,000 square feet it was not a suitable pillar. If the same area had been left in a circular form the shaft would have been perfectly safe. An oblong pillar was not a suitable pillar for a shaft, whatever it might be for buildings. His experience was also that the efficiency of a pillar was greatly impaired if the packing was not well attended to when the seam was first opened out. It was of no good putting in gate-end packs only; the packs should be built in the goaves at proper intervals, and if there was not sufficient rock from the ripping it must be obtained from the back of the goaves. In Lancashire the mines were steep, and pillars were left at right-angles to the seam, so that the pillar left to support a building was often shown on the plan at a distance from the buildings which it was intended to support. The pillars seemed to have no connexion on the plan with the buildings. All pillars should be placed at right-angles to the dip of the seam, and the larger area in the mine must be on the rise side of the buildings which were to be supported. If the pits were square, and not circular, they would have far more trouble with shafts giving way than they had now with circular pits.

When sinking, bricks were generally in great demand, and anything was considered good enough to send down the pit to back-fill. Some years ago a curious case arose in the Midland counties, where a colliery owner decided to sink a pit close to the edge of another property, and gave the adjoining colliery-owner notice that he required support for his shaft. He believed that in law he was correct, and the case, although undoubtedly unfair, was settled out of court. He thought they would see that the law of support was rather a delicate instrument, and certainly one that sometimes operated very unfairly. The owner of the surface took every precaution to look after himself, and the man who wrought the minerals was supposed to know what he was about, and look after himself also. He certainly thought it was a little unfair, whilst on this question of surface damage, that the landlord who received the value of his coal should, at the same time, make the lessees responsible for all damage to the surface. These were the ordinary conditions of a mining lease, and it seemed to him unreasonable, when they paid the landlord three times the value of the surface for the mines worked, that they should be made responsible for all damage to the buildings, surface-drains, etc. He was not a radical, and did not believe that the mines were national property, but there were some things which the landlords ought to do, and one was that when they received full value for their coal they ought to take some responsibility on the surface. No more accurate test of surface-subsidence could be made than by a canal, and in Lancashire there was one that had been raised, from time to time, 20 feet, through the coal having been wrought from under it. At a colliery in Yorkshire, damage arose to some property about 300 feet above the seam, but, in this case, there was clear evidence that an intermediate seam had been worked many years before, and that no damage would have resulted if the upper seam had not been previously worked. To avoid the glorious uncertainties of the law, the colliery-owner paid—as usual. A case came before him some time ago at Pleasley. A shaft was sunk through the Permian measures, and a fissure, 2 feet in width, was visible at the surface. It had an important bearing on this subject, for in many cases where colliery-owners were supposed to have caused damage they had nothing whatever to do with it. This fissure allowed the surface to fall, and it had caused the engine-houses and boiler-seatings to be disturbed, but happily it did not go in the direction of the chimneys. The only way in which they could deal with fissures was to fill in the spaces by brick arches well set into the sides and build on the top. They were found very largely in the Magnesian Limestone.

Another subsidence, the result of an earthquake about seventy years ago, was that of the Skegby church-tower. It was situated on the Magnesian Limestone, and had nearly fallen down, although no coal had been worked anywhere adjacent. If coal had been worked, it would have been alleged that the damage was caused by coal-workings. In cases of this sort, it was advisable to have a professional report on the defective structure before allowing the workings to approach it, and this evidence could be referred to afterwards. In this case, he took an architect's opinion on the tower some time before they applied to the Ecclesiastical Commissioners for England for a lease of the coal. When leases were being drafted colliery-owners should be careful, if they were obliged to covenant to make good any damage to the surface or buildings, to take power to enter and repair, and see that a subsidence-clause was inserted. There was no necessity for him to remind the members of a recent case, where a lessor had granted the lease of the coal, but as there was no subsidence-clause the working of the mine had been stopped. Houses that had been added to were always more likely to be damaged than those which stood as they were originally built. The subsidence generally took the shape of a curve, so that the top of the building opened first, and after a time the crack closed, and, except for appearances, pointing would make good all damage. He remembered a case where the workings approached a long wall, and caused the wall to fall away from the goaf, and buttresses had to be built on the off side of the wall. He was glad to see that Mr. Coke had already referred to the fact that coal was being abstracted under Newstead Abbey, and most careful observations were being made in connexion with that working. It was very desirable that Mr. Bagnold Smith should give the Institution a paper on the subject. Mr. Smallman had stated that the shrinkage of a 6 feet seam was 3 feet, and that the working of an 18 feet seam had caused a subsidence of 6 to 7 feet. He thought that there was no comparison between the two.

Mr. SMALLMAN, in reply to Mr. Longden, said that at Bedworth colliery they did not open out and work away from the shaft as in Derbyshire longwall. They drove to the extremity and worked back. The first break meant the same thing as the first subsidence which affected the surface, and it came on as the workings advanced towards the basset from the opening on the deep side. This might occur at any width between 90 feet in shallow, to 300 feet or more in deep mines.

Mr. LONGDEN asked whether Mr. Smallman agreed with the theory of the curved line of subsidence.

Mr. SMALLMAN rejoined that the subsidence on the surface, when the strata were flexible, was in the form of a curve, but where there was hard rock the ground was broken abruptly at the first break, leaving ridges from a few inches to 3 feet high, according to the thickness of the seam; afterwards the ground following the face of work subsided gradually in the form of a curve, having a full depression of 6 or 7 feet (as in the case mentioned where the first subsidence took place), and gradually curving to the line where the ground was only just beginning to be affected. This curve must be continually changing as the workings advanced until the surface had acquired the full subsidence. For instance, where workings in a 10 feet coal-seam were approaching a three-storied farm-house from the deep, where the strata near the surface were favourable to gradual subsidence, levellings were taken before the workings approached for a long distance, and the surface was found to sink 4 feet. The house, which was built parallel with the level, was firmly secured together by pins and side-plates; as the workings approached it was pulled over considerably, the top inclining to the deep, but as the workings gradually advanced it resumed its former position, and was left practically uninjured. The farm buildings, however, which extended from basset to deep, at right-angles to the house, and across the lines of fracture, were broken and damaged, because the long lines of brickwork could not follow the curve.

Mr. G. H. ASHWIN (Bedworth) said that he had tested the fact of the curve by putting in pegs and taking measurements before opening out from the rib. At a later date, he found that the distances had increased by quite 1 per cent. in a distance of a little over 1,000 feet in length, proving that a curve had taken place in the surface.

Mr. M. DEACON (Blackwell collieries) wrote that information on the subject of surface-damage was especially valuable to mining engineers. He had adopted the plan for some years of keeping an account of the effects of working coal under buildings, railways, etc., and if every colliery-manager would do the same, recording the variations in level of the surface, and the effects of the workings upon buildings, in a few years' time a mass of very valuable information would be collected. He agreed with Mr. Spencer's opinion that it was seldom profitable to a coal-owner to leave a pillar of coal to support a building, unless the building was one of special importance. As a general rule, there could be no doubt that working out of the whole of the coal was, in most cases, the most economical. The extent of damage to buildings could be sometimes limited by strapping the outer walls together, either by means of wrought-iron straps going round the building, or by means of tie-rods passing from

one wall to another. This was especially true in the case of large buildings, without tie-walls, such as factories, engine-sheds, public halls, etc. He thought that the liabilities which lessees took upon themselves for surface-damage in their leases very often amount to an unfair share of the risks. Further, that no lessee ought to be made liable for damage to buildings which did not exist at the date of the lease, and that if he was liable for existing buildings there should be a clause empowering him to enter and repair them in lieu of paying compensation.

Mr. G. S. Bragge (Swadlincote) said that the effects of working several seams one after another were apparently very erratic. The following illustrations of this might be of interest to the members:—In 1873-74, a 5 feet coal-seam, at a depth of 690 feet, was worked under a house, leaving a large area of the same seam unworked in the neighbourhood. The workings were well packed. No damages were claimed at the time. In 1886, the area left unworked in 1873-74 was worked, and caused subsidence and cracking of the house, for which damages were paid in 1887. In 1889-90, a 7½ feet coal-seam, 900 feet deep, was worked under the same area, and no damages were claimed: this seam was badly packed and gobbed. In 1893-94, a 3¾ feet coal-seam, 1,170 feet deep, was worked under the same area, and no damages were claimed: the seam was moderately well packed and gobbed. There are no faults in the neighbourhood, and the seams are practically level. In 1874-75, a 5 feet coal-seam was worked under a house, and no damage was effected. In 1885-86, a 7½ feet coal-seam was worked in the same neighbourhood, a pillar of 120 to 150 feet being left on each side of the house for its support, and the coal was worked on both sides of the pillar simultaneously. The house, however, was somewhat cracked and damaged. In 1894, the working of a 3¾ feet seam at a distance of 120 to 180 feet from the building (on one side) aggravated the damages of 1885-86, and caused serious injury to the property, for which compensation had been paid. There are two faults in the neighbourhood, each about 240 feet from the buildings. It was now believed that it would have been better to have left no pillar in the middle seam. The fact that a small pillar is worse than none was shown in another case. In 1853-54, a 5 feet seam was worked under a certain area, and in 1857 a 7½ feet seam was worked under a part of the same area. In 1859-60, a manufactory comprising a large number of buildings was erected on the above area. In 1875, the 5 feet seam, and in 1878, the 7½ feet seam, were again worked in the neighbourhood, leaving an irregular barrier of from 60 to 180 feet from the buildings, which, with the coal existing unde

the same, was supposed to be sufficient support for the surface. As the workings receded from the buildings, however, these began to show signs of subsidence, and heavy damages were claimed and paid. In this case a large fault ran close to the buildings, and was no doubt a factor in the disturbance. Subsequent workings (in the 3¾ feet seam) have not approached this area. Another case confirmed what Mr. Sebastian Smith said at a former discussion of the subject. A claim was made for damages to a house when the nearest coal-workings were 1,050 feet distant. Examination showed that the damages were due to the cracking and shrinking of the clay under the foundations of the house.

Mr. T. A. Southern (Derby) said that, as a reference had been made to Callon's Law, it might be useful to mention that the effects of subsidence and the sizes and position of pillars for the support of buildings were very ably treated and illustrated in Prof. Callon's *Lectures on Mining*.* Whilst listening to Mr. Smallman's remarks it appeared to him that several of the cases mentioned agreed exactly with the views put forward by Prof. Callon. The main theory enunciated by Prof. Callon, and to which he indicated certain exceptions, might perhaps with fairness be briefly summarized by the statement that the depression or subsidence of the overlying strata, caused by the working of any large area of a coal-seam by the longwall system, produced a fracture all round the limits or boundaries at which the excavation was discontinued, and that, except in very hard or very soft strata, this fracture was always propagated in a direction at right-angles to the plane of stratification of each stratum through which it passed. Consequently, the pillar required in a coal-seam to support a portion of the surface should be determined by projecting the outline of the surface to be supported down to the coal-seam, in a direction at right-angles to the plane of stratification of the intervening strata. The pillar must not, therefore, be placed vertically beneath the surface to be supported, unless the intervening strata were horizontal.

Mr. H. R. Hewitt (Derby) said he knew of a case where a coal-seam 5 feet thick was recently worked under a large town. The working-places were driven 30 feet wide, leaving 60 feet of coal, and the working-places were put through 36 feet wide to each other every 60 feet. In this way about one-half of the area of coal was extracted, and up to the present time no claim for surface-damages had arisen, although the depth

* *Lectures on Mining delivered at the School of Mines, Paris, by J. Callon, Inspector General of Mines.* Translated, at the author's request, by Prof. C. Le Neve Foster, D.Sc., and Mr. W. Galloway, H.M. Inspectors of Mines, 1876.

was only 600 feet. In preference to leaving a small pillar, it was frequently better to work the coal completely out as rapidly as possible, and to pay particular attention to the thorough packing of the stalls.

Mr. G. J. BINNS (Burton-on-Trent) said that there was one point he would like to mention, although it was really not germane to the discussion. It did not refer to buildings above, but to seams that had been worked in proximity to subjacent seams. Perhaps some member could tell him why after working overlying seams coal below should be "set," as the miners say. It could not be on account of the lack of support. Was it owing to the arch of the earth's surface—the sphericity of the earth's crust—being broken?

Mr. LONGDEN thought that it was the gas that had exuded which set the seam and rendered it hard to work.

Mr. BINNS suggested that the seam might be non-gaseous—a "gasless" seam.

The PRESIDENT thought that the explanation of gas was the correct one. The seam was "winded," as they said in the North of England; after the gas had gone out the seam was, in certain cases, very much harder to work.

Mr. LONGDEN said that they had a strong illustration of this fact in the deep soft and deep hard coal-seams worked so largely in Derbyshire. The thickness of the strata between the seams varied from 35 to 75 feet, and it had always been a moot point as to which seam ought to be worked first. He had always understood that the set was due to the gas having escaped or exuded from the coal. The gas acted like a wedge, forcing the coal down, helping the holing, and thus without it the holing was more difficult.

Mr. H. R. HEWITT thought that by taking away the overlying seam the useful effect of a part of the weight was lost, so necessary in getting and easy working; in the case just mentioned, the weight was due to only 42 feet of superincumbent strata, whereas if the upper coal-seam had not been worked there would have been a considerably greater weight. He could not agree with the gas theory, as the deep hard coal-seam was particularly free from gas, both where the deep soft was unworked and where it had been removed. If the gas theory were the correct one, he would expect to find more gas in the deep hard coal-seam in the former case than in the latter: an expectation which was not borne out in practice.

The PRESIDENT said that the discussion had greatly added to the value of his paper, which had served as the nucleus for the information which

they had now obtained. He had received the following particulars kindly sent by a friend in Yorkshire. In one instance, by working a coal-seam, 2 feet 4 inches thick, at a depth of 540 feet, at the foot of a fault, a building was affected at a lateral distance of 210 feet, the building having been erected on the surface-line of the fault. This house was now entirely unfitted for habitation. In another case, the working of a coal-seam, almost horizontal, at a depth of 1,000 feet, lowered the land in a park and caused the banks of a brook to sink more than 2 feet. This was proved by actual levelling, and there was only an interval of ten months between the working of the coal to the sinking of the land, although there was no alluvial deposit. These circumstances led to the impression that there must be old intervening workings, of which there was no record. In another case, a coal-seam, averaging 5 feet in thickness, at a depth of 240 feet, was worked; here the subsidence was about 3 feet, and it took approximately two years for this subsidence to be completed after the coal had been worked. The old workings of the high main coal-seam in Northumberland, 600 feet deep and 6 feet thick, are filled with water. For thirty years, the low main coal-seam, 6 feet thick and 390 feet deeper, has been entirely removed, but no water has come down. Two other coal-seams below the low main coal-seam are also being entirely removed. He (the President) proposed that the discussion be now closed.

Mr. W. H. Hepplewhite (Nottingham) wrote that during the discussion, he had not heard the experience of anyone partially leaving an upper and totally extracting a lower seam under a large number of buildings. The following facts had come under his notice some years back. An upper seam, 5 feet thick, 435 feet from the surface, was partially worked below a number of workmen's cottages, in fact, a small village, the approximate area being about 8 acres, without any perceptible lowering of, or damage to, the property. It would be seen from Fig. 1 that from 25 to 30 per cent. of the coal was worked, under the buildings, in long strips varying in width from 20 to 30 feet, with corresponding pillars of solid coal varying from 33 to 90 feet in width. It would be observed that the coal worked and the pillars left were placed almost at right-angles to the rows of cottages. A few years afterwards a lower seam, 7 feet thick, at a depth of 1,092 feet from the surface and 657 feet below the upper seam, was worked on the longwall principle, under the cottages, and the whole of the coal extracted. The working-faces approached the pillars in the upper seam in a diagonal direction, and no damage to the cottages was observed until nearly the whole of the coal

had been extracted. Then the corner of the first building approached began to crack, and the occupants had to be removed, until ultimately the building toppled over. Many others followed in like manner, and ere the general subsidence had been completed heavy damage was done to the property. It was subsequently found that the whole of the buildings had sunk about 4 feet. He was of opinion that had solid coal

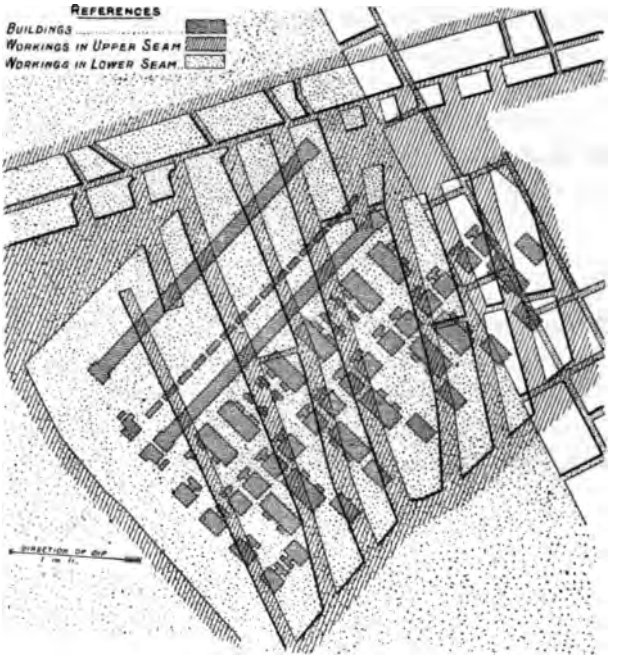


Fig. 1. Scale, 4 Chains to 1 Inch.

been left in the upper coal-seam, the whole of the lower seam might have been extracted without any appreciable damage to the buildings; because, undoubtedly, the pillars left in the upper coal-seam did not settle uniformly with the surrounding gobbing, but when once set on the move, the pillars continued longer on the "creep," and prevented the buildings from settling regularly.

Mr. LONGDEN moved that a vote of thanks be given to Mr. Spencer for his interesting paper.

Mr. C. R. MORGAN seconded the resolution, which was agreed to.

The PRESIDENT, in acknowledging the vote of thanks, said that he was indebted to Mr. T. Douglas and Mr. A. L. Steavenson, two very old friends, who had contributed a large amount of the information

contained in his paper. He had asked a number of other gentlemen, but in consequence of the glorious uncertainty of the law, they had fought rather shy of supplying him with information. The discussion had added most materially to the value of his paper, and he felt much indebted to all those who had joined in it.

DISCUSSION UPON MR. DEACON'S PAPER ON "THE USE OF EXPANSION-GEAR AS APPLIED TO COLLIERY ENGINES."*

Mr. J. A. LONGDEN (Stanton-by-Dale) said that the Guinotte gear described in the paper was erected about twenty years ago, when he had charge of the Blackwell collieries, and at that time they ascertained that the saving by using this gear was one boiler in five or 20 per cent. Mr. Deacon stated that the boiler saving was 25 per cent. with the Thornewill & Warham cut-off, and he thought that the difference arose in this way. The boiler-pressure with the Guinotte gear was 55 lbs. per square inch, whilst with the new gear the steam-pressure was 80 to 100 lbs., and the economy at the higher pressure was naturally greater in the boiler-space required. With regard to the fan-engines, 30 inches in diameter, Mr. Deacon gives two indicator diagrams, one wire-drawn (Fig. 5, Plate XIX.), and the other with expansion-gear at work (Fig. 6, Plate XIX.). He (Mr. Longden) fifteen years ago made experiments with each for a week with the fan running at the same number of revolutions, one week with the wire-drawn engine, and the next with the expansion-gear in action. One Lancashire boiler, 28 feet long by 7 feet in diameter, was set apart for the purpose. The coal was weighed, but no account was taken of the evaporation of water. The result was as nearly the same in each case as possible. It was only fair to state that the steam-pressure at that time was 55 lbs. per square inch. He did not know all the expansion-gears described in Mr. Deacon's paper, but, speaking generally, the anticipated economy would not be obtained unless the steam-pressure was 100 lbs. per square inch. At Silverhill colliery, he had erected a Capell ventilator, which was driven by an old winding-engine, with two cylinders fitted with drop-valves. This was a false idea of economy, as the old engines should have been sold, and an efficient

* *Trans. Fed. Inst.*, vol. vii., page 672; and vol. viii., page 373.

engine properly constructed applied to the work. The cylinders were too large, consequently a very small useful effect was obtained, and an enormous quantity of steam was used. He applied the Melling expansion-gear, and a remarkable saving of steam of about 50 per cent. had been effected. The cylinders were too large for the present duty of the ventilator, although they would be the right size in the future. He thought the members would agree with him that one of the great causes of loss in working a ventilating-fan was owing to the use of steam-cylinders of too large area. A fan was erected capable of dealing with the final development of the mine. If 80,000 cubic feet of air were required at the present time, a fan would be erected capable of producing 150,000 cubic feet per minute, and the steam engine would be erected capable of running the fan at such a speed as to produce 150,000 cubic feet, and thus there was an enormous loss of power. Engine bed-plates could easily be made to take large and small-sized cylinders, and fan-engines should be erected with small cylinders, which could be replaced by large ones, as occasion required. It was certainly desirable to have some automatic expansion-gear, so as to regulate the consumption of steam to the amount of work the fan was performing at the time.

Mr. Alfred Davies (Burton-on-Trent) said that in the Thornewill & Warham expansion-gear the cut-off was automatically adjusted in proportion to the amount of work to be performed, and this would account for the increased economy obtained as compared with the Guinotte gear, which did not comply with these conditions; further economy being due, as suggested by Mr. Longden, to the increased steam-pressure and to the correspondingly greater number of expansions of the steam. With regard to the experiments on the fan-engines carried out by Mr. Longden, and in which he obtained no benefit from the use of expansion-gear, it was evident that this was due to the cylinder being too large for the work to be done, and that a smaller cylinder with a moderate range of expansion would have given much better results, and that throttling the steam with the smaller cylinder would have then resulted in a loss.

The discussion was adjourned.

DISCUSSION UPON THE "REPORT OF THE PROCEEDINGS OF THE FLAMELESS EXPLOSIVES COMMITTEE. PART I.—AIR AND COMBUSTIBLE GASES."*

Mr. A. C. KAYLL (Newcastle-upon-Tyne) said that explosives formed one of the subjects of the day. Since 1888, many important and valuable explosives have been placed upon the market, each had in its turn made its *début* with a great flourish, and for each it had been claimed that it was perfectly safe when fired into an explosive gaseous medium. Each inventor had conducted experiments with the object of proving the superiority of his explosive to the detriment of its predecessors, consequently the advantage of an independent enquiry was sufficiently apparent. In 1888, the North of England Institute of Mining and Mechanical Engineers decided upon making the enquiry, and, in 1893, the experiments were commenced. The experiments were made with the so-called flameless or safety explosives:—Ammonite, ardeer powder, bellite, carbonite, roburite, and securite. The tests were made in mixtures of coal-gas and of pit-gas with air, with shots stemmed and unstemmed. All the shots were fired by means of a detonator and a magneto-exploder. Briefly, the trials proved that all the explosives showed flame. In coal-gas, the explosives, both when stemmed and unstemmed, proved capable of igniting the gaseous mixture; in pit-gas, none of the explosives when stemmed fired the gas, twenty experiments being made in each case. When unstemmed, ardeer powder, ammonite, and securite ignited the mixture, but the others did not. There was a tabulated list printed at the end of the *Report* of every experiment that had been made throughout the enquiry, with the exception of those made with common powder, and the extensive series that had been made by a sub-committee in their preliminary work. The conclusions of the committee were as follows:—

1.—All the high explosives (ammonite, ardeer power, bellite, carbonite, roburite, and securite) are less liable than blasting-powder to ignite inflammable mixtures of air and fire-damp. These explosives, however, cannot be relied upon as ensuring absolute safety when used at places where inflammable mixtures of air and fire-damp may be present.

2.—The variable results following upon the detonation of high explosives appear to be due in some measure to defective admixture of, or variation in, the proportions of the ingredients used in the manufacture of the explosive. In view of the changes from time to time made in the proportions and constituents of high explosives, it seems desirable that this information should be afforded by the manufacturers to the users of the explosive.

* *Trans. Fed. Inst.*, vol. viii., pages 227 and 593.

3.—In the storage of high explosives, it is desirable that every care should be taken to ensure their being maintained in a proper condition. It is also certain that these explosives alter in character with age.

4.—It is essential that similar examinations of the working-places and precautions which are in force in mines where blasting-powder is used, should be rigidly observed when a high explosive is employed.

5.—In selecting a high explosive for use in a mine, it should not be forgotten that the risk of explosion is only lessened and not abolished by its use.

6.—All of the high explosives on detonation produce evident flame.

7.—The emission of flame from a blown-out shot of a detonated high explosive is not prevented by the quantity or length of stemming used.

8.—In the case of a charge of a high explosive which has missed fire, if a short length of stemming (proved up to 8 inches) has been employed, the charge can be detonated by another cartridge of the explosive and additional stemming being placed in the hole, in front of the original stemming.*

The PRESIDENT said that in No. 6 conclusion it was stated that "all of the high explosives on detonation produce evident flame." Did that mean without any stemming?

Mr. KAYLL said that flame was observed both with stemmed and unstemmed shots. There was very little evidence of flame when the shots were stemmed; but in every case, except those instances shown in the tables, flame was seen with all the explosives when unstemmed.

Mr. H. R. HEWITT (Derby) said that the *Report* stated that some of the explosives, although not absolutely flameless, were nearly safe when properly stemmed, but they had always to prepare for blown-out shots, and in that case, he believed there was evident flame shown; so much so that the Committee did not recommend any explosive. The lengths of the flames seemed to be extraordinary, some as long as 16 feet in the lower percentage gaseous mixtures, and he would like to know whether there was ignition of the gaseous mixtures by those flames. He also enquired whether the 10 per cent. gaseous mixture was exploded at the same time as the flame was seen, and whether the flame was due partly to the ignition of the gaseous mixture? If the flames did not ignite the gas it was for the same reason that when gunpowder was placed on gun-cotton, and the cotton lighted, the gun-powder was left unconsumed, because of the quickness of the resulting flame. He would like Mr. Kayll to explain why there was an interval between the shot and the explosion, in some cases as much as 1½ seconds, as he was at a loss for an explanation.

Mr. T. A. SOUTHERN (Derby) said the members would find that the *Report* was full of interest. The cases of delayed ignition were very remarkable. Could Mr. Kayll give further information with reference to this point?

* *Report*, page 40.

The PRESIDENT, referring to the experiments without stemming in pit-gas, said that with bellite there was no ignition in twenty-five shots, but with securite there was one ignition in one shot. Did that mean that only one shot of securite was fired? With ammonite there was one ignition in one shot; roburite, no ignitions in twenty-five shots; carbonite, no ignitions in twenty-five shots; and ardeer powder, one ignition in four shots. These experiments seemed to point to very much greater freedom from flame in the case of three of the explosives than with the other three. With the exception of ammonite, ardeer powder, and securite the explosives were almost absolutely flameless.*

Mr. KAYLL said that the Committee decided upon a stated number of trials for each series, and when an explosive ignited the mixture further trials with it were discontinued. Their programme in pit-gas mixtures was to fire twenty-five shots, unless they obtained an ignition with a smaller number of experiments. For instance, bellite passed through twenty-five trials when unstemmed without igniting an explosive mixture of pit-gas and air.

The PRESIDENT—It might have fired twenty-five times?

Mr. KAYLL—No; if it had fired the mixture at the first shot, further trials with it would not have been made.

The PRESIDENT asked whether similar results had been obtained with ammonite?

Mr. KAYLL replied that was so, when the explosive was stemmed. Ardeer powder showed much less flame, it was very safe in coal-gas and air, but when tested in pit-gas it ignited the gaseous mixture. In answer to the other questions, he (Mr. Kayll) said that the flames were elongated when fired into mixtures containing 6 per cent. and less of gas. With 10 per cent. mixtures, the ignition occurred simultaneously with the detonation of the shot, except in the few instances mentioned in the *Report*. The following was the method adopted for obtaining gaseous mixtures:—One minute was devoted to running the gas into the chamber, the fan was run one minute for the purpose of mixing the gases, and another minute usually elapsed before the shot was fired. It had been found by numerous experiments that the mixture was thoroughly uniform in its contents. The gaseous mixture had often been kept in the chamber for four or five minutes, when it had not been ignited by the shot, and it had then been ignited with a match, and it was always found that the mixture was violently explosive. There was a great difference in the force and character of

* *Report*, page 19.

the explosion according to the position in the chamber of the point where the mixture was ignited. Mr. Hewitt had drawn attention to the question of "delayed ignitions," which was a very interesting matter, and had not hitherto been satisfactorily explained.

The PRESIDENT moved that a hearty vote of thanks be accorded to Mr. Kayll for his explanations.

The resolution was carried by acclamation.

The discussion was then adjourned.

DISCUSSION ON MR. JOHN MORISON'S PAPER ON "WALLING AND SINKING SIMULTANEOUSLY WITH THE GALLOWAY SCAFFOLD."*

Mr. H. STEVENSON (Portland collieries) wrote that a detaching-hook had not been used with the Galloway scaffold at the Newbattle collieries, and this in itself was a serious defect. It was to be regretted that no details of the time occupied in sinking and bricking had been recorded in the paper. The slinging of the scaffold appeared to be defective, as, although there were four points of attachment at the surface, there were only two at the scaffold, and the point of support was at the axles, which were free to swing laterally. The ropes, unless of special make, were too large for the pulleys, and the load left no margin for sudden or extra weight and wear of the ropes by the riders. There was also an absence of any secondary or "catch" provision to the scaffold. Did the sinking require quick winding, and, if so, why? When bricking a 240 feet length, the scaffold, say, being 234 feet up, and the sinking at the time, say, 39 feet down, in all 273 feet, what was done with the kibble? When starting away from the bottom, if steadied, then the time spent from when the engineman slowed until he lifted the rider was time lost. The writer could not agree with Mr. Morison that folding-doors over a sinking-shaft were the safest or best for dealing with materials out of, or to be sent into, the shaft. For a shaft 20 feet in diameter, three bricklayers was a very small number to be at work, and it would be misleading to base any calculations on such data. Owing to the brick-kibble working to only one side, the materials for three-fourths of the shaft must be carried, instead of tipped at the place required. The cost of a chargeman for so short a length of work

* *Trans. Fed. Inst.*, vol. viii., page 118.

done could not be called economical, and to this might be added the time lost in bringing up the sinkers when timber was being withdrawn from bad ground, and in drawing the bricking shift for shot-firing. Mr. Morison could not be congratulated upon the quality of work done by the "skilled bricklayers," or there would not have been any tight places in the shaft. The space through which the two kibbles worked took up a quarter of the area of the shaft area, at a point where the greater number of the workmen were usually employed. The timbering, if correctly drawn in Figs. 1 and 2 (Plate VIII.), would lead to the impression that the pit had been sunk upwards, the temporary cribs only giving support to the ground by the size of the wedge, the polling-boards coming between the wedge and the crib, thus giving no support to the sides of the shaft. He (Mr. Stevenson) preferred the use of wrought-iron bars, curved to the diameter of the shaft, and coupled together by two or more bolts, which, by a match-up piece, could be adapted to any irregularity of the ground. He had often inserted one of these rings, 17 feet in diameter, together with the polling-boards, in less than half an hour.

Mr. JOHN MORISON (Newcastle-upon-Tyne) wrote that there was nothing in the Galloway scaffold to prevent the use of detaching hooks if so desired. The writer failed to follow Mr. Stevenson when he says that "the slinging of the scaffold appeared to be defective," etc. Plate VIII. showed clearly four points of attachment to four pulleys on the scaffold, and not at two points, as Mr. Stevenson alleged. The points of support must clearly be the four pulley axles which are not free to swing laterally, except with the body of the scaffold. It is hardly necessary to explain that the ropes used were of suitable make for the pulleys. Mr. Stevenson says that "the load left no margin." As a matter of fact, the margin was nearly 20 to 1, viz., breaking strain of rope, 396 tons, and load 20 tons. There was not, so far as the writer could judge, any need for the "secondary or catch provisions" to be provided, as suggested by Mr. Stevenson. The sinking did require quick winding, it being essential in this, as in all other sinkings, to remove the *débris* as rapidly as possible after a blast. In the circumstances mentioned by Mr. Stevenson, when the scaffold might be 273 feet from the bottom, the kibble was steadied from the bottom, as is usual under ordinary circumstances, and no more time was lost than usual. As stated in the paper, the kibble was slowed in approaching the rider, but not as Mr. Stevenson suggested, from the bottom up to the rider. The engineman merely slowed for a few feet in approaching the rider, and in doing so lost very little time. The question

of folding-doors or other arrangements over the sinking shaft was a matter of opinion, and the merit of either one system or another was dependent upon the constructive and working details. The writer in giving the data upon which Mr. Stevenson says it would be "misleading to base any calculations," gave the data which actually existed. He (Mr. Morison) thought that the proportion of 3 labourers and 1 sinker in charge, who also laboured, to 3 bricklayers was economical. Very little time was lost owing to the other causes mentioned. About $\frac{3}{4}$ inch, as explained in the paper, was allowed between the scaffold and the brickwork, and the loose wing described was necessary in order to clear pipes, pumps, buntons, etc., these being the tight places for which provision was made. Mr. Stevenson went out of his way to conclude that the brickwork was tight or defective, and was quite unwarranted in this conclusion. The timbering as shown in Plate VIII. was correctly drawn, the backing deals being put in sideways and supported by backing where necessary, as was usual. Consequently Mr. Stevenson's conclusion was wholly baseless and incorrect. The use of the Galloway scaffold would not prevent Mr. Stevenson from using wrought-iron bars in preference to wooden cribs should he desire to use them.

The meeting was then closed.

THE MINING INSTITUTE OF SCOTLAND.

ANNUAL GENERAL MEETING,

HELD IN THE HALL OF THE INSTITUTE, HAMILTON, APRIL 11TH, 1895.

MR. GEORGE A. MITCHELL, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and confirmed.

The SECRETARY read the report of the Council as follows:—

COUNCIL'S REPORT.

In submitting this, the seventeenth annual report, the Council have pleasure in stating that substantial progress has been made during the past year.

The history of the Institute shows continuous increase in the number of members from year to year, the only exception of importance being in 1893, when the Institute joined The Federated Institution of Mining Engineers; but it is satisfactory to record that the interruption then experienced has been surmounted, and that the membership is again on the increase.

At the date of last annual meeting there were:—

On the roll		481
Added during the year		37
		518
Died	9	
Retired	9	
Cut off on account of non-payment of subscriptions	13	
	—	31
On the roll at this date		487

of whom 387 are federated and 100 are non-federated.

The volumes of the *Transactions* received by federated members contain the proceedings of seven institutes, while non-federated members receive only the proceedings of this Institute. The Council recommend to non-federated members the desirability of federation, so as to obtain the benefit of the whole *Transactions*. During the last year the *Transactions* have not been issued with the regularity to which this Institute has been accustomed, and some discussions have had to be delayed in consequence. This is a state of things which the Council are satisfied you will consider undesirable and inconvenient. The printing being under the management of the Council of The Federated Institution, this Council cannot take the matter in hand direct; but all that can be done will be done to ensure timely issue of the *Transactions*.

The abstract of accounts which follows shows that for the past year the expenditure has exceeded the income. This was not anticipated at the beginning of the year, but is accounted for chiefly by the fact that the expenses of The Federated Institution required a payment per member of 2s. in excess of that of the preceding year, as well as a call of 1s. 6d. per member to make up a deficit of that year.

The papers read during the year and published in the *Transactions* are as follows :—

President's Address on "The Commercial Aspect of Coal-mining."
"The Coal-fields of Labuan."
"The Ross Rock-drill."
"Historical Notes on Early Plans for Coal-washing."
"Remarks on Winding Engines."
"The Barclay Crown-valve Exhaust-gear."
"An Overhanging Beam Pumping Engine."
"Water-tube Boilers."
"The Design, Efficiency, and Application of Electric-motors for Transmission of Power in Mines."

The annual excursion in September was to Newcastle-upon-Tyne, coincident with a meeting of The Federated Institution of Mining Engineers in that town. The number of members taking part in the excursion was smaller than usual.

The Council a few months ago appointed a Committee to make investigations and report on the best systems of mine-drainage. By the appointment of this Committee the Council have set before themselves an undertaking of some importance, involving a great amount of work on the part of the members of the Committee. It is confidently hoped that the results will be of value to the Institute, and the Committee rely on the active assistance of the members in this matter.

The donations received during the year were as follows:—

Donor.	Donation.
J. T. Robson	Inspector's Report, South Wales District, 1893. Summaries of Statistics, 1893. Report of Albion Colliery Explosion.
W. J. Potts	Potts' Mining Register, 1894-5.
J. B. Dalzell	A collection of 27 books and pamphlets relating to mining.
B. T. A. Bell	Canadian Mining Manual, 1894.
Ralph Moore	A sum of £3 3s. for books.
J. M. Ronaldson	Royal Commission's Report on Labour, 15 volumes. Mines Inspectors' Reports, 1893. Mineral Statistics, etc., 6 volumes.
Wm. Williamson... ...	Mining Notes and Formulæ, 1 volume. Pamphlet: Examination Questions.
North of England Institute of Mining and Mechanical Engineers	Report of the Proceedings of the Flameless Explosives Committee, 1 volume.
James Ashworth... ...	Pamphlet: A Review of the Coal-dust Question.

The societies and technical papers with which an exchange is made now number 47, and new books are being added to the library from time to time by purchase. These, together, provide regular and fresh information as to the progress of mining and allied industries in all parts of the world.

It is with regret that the Council have to record the death of one of their members, Mr. Wm. Archibald, who has held office for some years, and whose name stands on the list for re-election.

There have been nine meetings of the Council during the year.

The adoption of the report was agreed to unanimously.

The Secretary read the following abstract of the Treasurer's accounts for the year, duly audited:—

The Treasurer in account with the Mining Institute of Scotland

For the Session 1894-95

Receipts	£	s.	d.
To Balance (1893-94)	153	19	4½
„ Subscriptions, 2 Federated Life Members	25	4	0
„ „ 1 „ Member (1893-94) at 21s.	1	1	0
„ „ 1 „ „ (1894-95) at 30s. 6d.	1	10	6
„ „ 163 „ Members „ at 31s. 6d.	256	14	6
„ „ 219 „ „ „ at 21s.	229	19	0
„ „ 37 Non-Federated „ „ at 21s.	38	17	0
„ „ 65 „ „ „ at 10s. 6d.	34	2	6
„ „ 3 Federated „ (1895-96) at 31s. 6d.	4	14	6
„ „ 1 „ Member „ at 21s.	1	1	0
„ Transactions, etc., sold	4	10	11
„ Bank Interest	2	8	0
„ Rents of Hall	5	2	6
„ Donation	5	5	0
„ Arrears for Session 1893-94	0	10	6
„ Federated Institution of Mining Engineers	10	6	3
	£775	5	8½

Payments	£	s.	d.	£	s.	d.
By Federated Institution of Mining Engineers				146	4	3
„ Printing and Stationery				30	18	3
„ Books and Bookbinding				22	14	3
„ Rent of Halls				10	3	11
„ Tax and Assessments				2	14	3
„ Stamps, Carriages, etc.				10	10	8½
„ Sundry Payments				2	18	8½
„ Expenses of Council, etc.				20	1	11
„ Donation				5	14	11
„ Salaries				31	14	11
„ Balance in Bank	141	7	1			
„ „ in Treasurer's Hands	11	14	8½			
				153	19	4½
				£775	5	8½

April 8th, 1895. Examined, compared with vouchers, and found correct.

J. H. DALZELL,
JAMES A. MILLARKEY, } Auditors.

The following gentlemen were elected by ballot:—

FEDERATED MEMBERS—

Mr. ROBERT W. B. CREEKE, Durie Foundry, Leven, Fife.
Mr. JAMES RAMSAY, Kenmuirhill Colliery, Broomhouse.
Mr. WILLIAM RAMSAY, Enterkine Lodge, Annbank.

ELECTION OF OFFICE-BEARERS.

The PRESIDENT declared the following office-bearers duly elected for the session 1895-96:—

PRESIDENT.

Mr. GEORGE A. MITCHELL, 63, Bath Street, Glasgow.

VICE-PRESIDENTS.

Mr. JAMES HASTIE, Greenfield Colliery, Hamilton.
Mr. WALLACE THORNEYCROFT, Merryton Colliery, Hamilton.
Mr. JAMES T. FORGIE, Bothwell Collieries, Bothwell.
Mr. H. M. CADELL, Grange, Bo'ness.

COUNCILLORS.

Mr. THOMAS ARNOTT, Fernbank, Newton.
Mr. DUGALD BAIRD, Comely Bank, Leven.
Mr. ALEXANDER FAULDS, Calderbank, Airdrie.
Mr. JAMES FREW, Dunaskin, Ayr.
Mr. ALEXANDER M. GRANT, Gardenhill, Kilmarnock.
Mr. JAMES HAMILTON, 208, St. Vincent Street, Glasgow.
Mr. DOUGLAS JACKSON, Coltness Iron Works, Newmains.
Mr. JOHN MENZIES, Auchinraith Colliery, Blantyre.
Mr. MICHAEL ROSS, Westburn Colliery, Newton.
Mr. FRED. J. ROWAN, 121, West Regent Street, Glasgow.
Mr. JAMES WYPER, Townlands Colliery, Hamilton.

Mr. JAMES WALLACE, Wester Gartshore colliery, Kirkintilloch, was nominated for election as a Councillor in room of Mr. Wm. Archibald, deceased.

The PRESIDENT said that he thanked the members for the honour they had done him in again electing him as President. He had been more than ever impressed during the past year with the important place which their Institute filled. The meetings gave valuable opportunities for the discussion of the many and varied problems which were of interest and importance to those engaged in mining, and they all desired that mining men in the various parts of Scotland should realize this, and should take such part in the work of the Institute as they might feel qualified for. He need hardly say that papers were not the only things wanted, but that the discussions were very often the most valuable part of the proceedings, and members would add greatly to the interest of the meetings if they

contributed more largely to the discussions; and those members who could not be present should endeavour to send written communications when any subjects were discussed in which they were interested. It had been thought by some that available subjects would become exhausted, and that there would be difficulty in keeping up the interest in the meetings. He had no such fear. There were many subjects of importance which had often been brought forward, but which were still far from exhausted; there were many subjects which had hardly been mentioned; the march of progress would supply others as they were required. As an example of an old subject which had many times been before the members, he might mention that of mine-drainage. It was by no means exhausted, and every year it was becoming of greater importance as shafts became deeper and districts became more heavily watered. The council had recognized that there was still something to be said on the question by the appointment during the past year of a pumping committee. This committee had entered upon a most important investigation, and he anticipated that the results of its labours would be most valuable even if it did little more than tabulate, for reference, any results arrived at. Another well-known subject which was deserving of more attention than it had hitherto received, was that of the geology of the coal-fields, and it was to be hoped that there might be many contributions in the future that would add to the knowledge of the various districts. Among the newer subjects which were engaging attention, there was none more interesting than the use of electricity for the transmission of power. They had had several papers during the past few years, which had described electrical installations for mining purposes, and members would recognize from the transactions of other institutes, that more and more notice was being taken of this power every year. Five or six years ago, electricity was looked upon by mining engineers as something mysterious, to be avoided as dangerous and unreliable, but now at many collieries electric pumps and haulages were in satisfactory operation. Considerable extension of the use of this power in the future was probable. It might be, for instance, that motors would be very largely used for haulage purposes. They could be so easily applied at any part of a mine, that it might be possible to make use of them for working branch roads and for driving light ropes as feeders to the main haulages. There was a probability, too, that the power would be, in the future, largely used for working coal-cutting machinery. As some members of the Institute would, no doubt, be aware, there had been in operation for some time an electric coal-cutting machine at Glenclelland colliery, and he intended that night to read a

description of this cutter. He believed that this paper would be of more interest to the members than any address which he could give on general subjects. Before he read the paper, he would refer to the death of Mr. William Archibald, who was nominated for election as a member of Council at the last meeting. Mr. Archibald was well-known as a capable colliery manager, and much liked by those who knew him as a man of upright character and genial and pleasant manner. By his sad and unlooked for death, their Institute had lost one of its most useful and interested members.

The PRESIDENT then read the following paper on "Electric Coal-cutting at Glenclelland colliery":—

ELECTRIC COAL-CUTTING AT GLENCLELLAND COLLIERY.

By GEORGE A. MITCHELL.

The writer resolved a few years ago to make the experiment of applying electricity to the working of coal-cutting machines at Glencleland colliery. This resolution was taken after careful examination by the writer and Mr. Thomas Dewar, manager of the colliery, of the different compressed-air machines working in Scotland, and with some hesitation, on account of the fact that for coal-cutting, electricity, so far as least as generally known, had never been successfully employed in this country. There was only one previous attempt for such an application of electricity made in Scotland—that by the late Mr. Durie at Elphingstone colliery. The machine was not successful, but its non-success was possibly due to the unsuitable nature of the seam to which it was applied. Some machines had been introduced in England, such as the Goodden bar-cutter machine, but, as far as could be ascertained, they were still in the experimental stage, and little information could be obtained concerning them.

There seemed no good reason, however, why the application of electricity to coal-cutting should not be as successful as its applications to haulage and pumping, for which purposes its use was rapidly extending, and the advantages of electricity over compressed-air for the transmission of the power required, if successfully applied, seemed undoubted.

As the experiment seemed worth making, after corresponding on the subject with various electrical firms, an arrangement was made with Messrs. Ernest Scott & Mountain, of Newcastle, to make a machine to the writer's instructions. They agreed to share the risk attached to the experiment, and in the construction of the machinery they spared no trouble to make everything as it should be.

The plant has been in operation for about nine months.

The writer thinks that it will be of interest to the members, in describing the machinery, to give at the same time some notes regarding the various points considered in its construction, and also some of the results of the experience in its working.

On the Surface.—The dynamo is of the improved type known as the Tyne dynamo, it is shunt-wound and constructed to give an output of 166 ampères at an electro-motive force of 300 volts, when running at a speed of 750 revolutions per minute. It is complete with sliding bed-plate, tightening screws, brackets, etc. This dynamo is to be converted into a compound-wound machine, as this is considered more suitable for the permanent arrangement.

What is necessary in an installation of this kind, where there is considerable variation in the power required, is to have the electro-motive force as nearly as possible constant for the same speed of engine, and independent of the amount of current required. This result is attained fairly well by the use of a shunt-wound dynamo when it is working well within its capacity, but when it is overloaded the electro-motive force rapidly falls, and, if the load be maintained, there is a possibility of the current falling to zero. A compound-wound dynamo may be so made that the electro-motive force is practically constant within the limits of certain current-variations. These variations for different dynamos are shown by their characteristic curves, which are to the dynamo what the indicator-diagram is to the steam engine. A description of these curves may be found in text-books on the subject.

The engine which drives the dynamo is horizontal, with a cylinder 18 inches in diameter and 3½ feet stroke, with scroll expansion-valve, and ball-governor, and works at about 50 revolutions per minute. The steam-pressure at the boilers is 45 pounds per square inch. The engine drives from a turned fly-wheel 11 feet in diameter, and there is a counter shaft with pulleys to bring the speed of the dynamo up to 750 revolutions per minute. The engine is connected to a counter which indicates the number of revolutions worked for each shift.

The switch-board, besides switches, has a voltmeter and an ampèremeter, and there is also in circuit a recording ampèremeter or ampèreograph, which indicates the variation of the current during each cut.

Cables.—There is at present in use a temporary cable of 19/18 wires, insulated with vulcanized indiarubber and braided, carried down the shaft and along the underground workings, on insulators, to the coal-face, a distance of about 2,200 feet.

The permanent cables, which are yet to be put in position, are (*a*) for the shaft 37/15 cable covered as follows, (1) one coat of pure Para india-rubber, (2) two coats of vulcanizing indiarubber, (3) one coat of vulcanizing tape, (4) the whole vulcanized together, (5) braided hemp, (6) armour of galvanized iron wires, and (7) braid and preservative com-

pound. The insulation-resistance is 1,000 megohms per mile, and t cable is of sufficient size to carry without excessive loss the whole curre which the dynamo is capable of producing. This cable will be attach at the bottom of the shaft by means of water-tight switches to (*b*) t smaller cables, which will each be large enough to carry the curre required for one coal-cutter. These cables have each 7 wires, No. 15 wi gauge, insulated with vulcanized indiarubber (the insulation-resistan being not less than 600 megohms per statute mile), then armoured wi one covering of galvanized iron wires, and braided and compounded ov all. These cables will be carried on insulators. The size is sufficient carry 30 ampères without undue loss. (*c*) Flexible concentric cabl insulated with vulcanized indiarubber, and spirally armoured outside, w be used at the coal-face.

Coal-cutter.—The first machine supplied, which, as stated, has be working about nine months, had certain slight defects, which are bei rectified in the second machine which is not delivered yet (Figs. 1 and Plate VI.). This machine is $7\frac{3}{4}$ feet in length, 40 inches in breadt and 23 inches in height. The frame is made of steel, and the tot weight is about $32\frac{1}{4}$ cwts., which is considerably lighter than the fir machine which has been in operation. There is a strong sheet-ir covering over the motor, and there are also bands over the toothe wheels to prevent breakage by falls from roof. The motor is serie wound, which seems to be as good as any form for the purpos The only disadvantage in this form of motor is that the speed becom rather high where there is no load, the revolutions increasing, as teste to 1,300 per minute. A shunt-wound motor would give a more consta speed, but it has not the same power for starting, although there a devices for overcoming this difficulty.

The speed for which the motor is designed is 600 to 700 revolutio per minute, and it gives 12 effective brake horse-power, although it w work up to a greater power if necessary.

The cutter (Fig. 2) is of the disc type, and is intended to combi the advantages of two well-known machines, which are worked Scotland by compressed air—the Rigg and Meiklejohn, and the Gillo and Copley machines. These machines differ from each other in sever important particulars. The Rigg machine has a disc which works the level of the pavement at the comparatively high speed of 60 or 70 more revolutions per minute, and which has generally only about s cutters in the circumference. The picks enter the cut in the front, an turn towards the back of the cut. The Gillott machine has a disc whic

cuts above the rail-level; the disc works at the slow speed of about 15 revolutions per minute, and it has about 20 cutters in the circumference. The disc turns from the back of the cut to the front, the aim apparently being to draw the machine into the coal-face rather than thrust it out from it. In working, this machine is slightly tilted up by raising the sleepers at the outside, and the disc slopes from a few inches above the pavement in front of the cut to the pavement at the back. The disadvantage of this arrangement is that if the pavement below the holing be hard, the picks are apt to be blunted by striking against it at the back of the cut, and if there be a coal-holing there is some waste of coal in the front, or if there be a fireclay holing there is the necessity of cleaning the fireclay from the coal at the back of the cut. The advantage is that the disc clears itself better of the cuttings than in the Rigg machine, as there is no obstruction to prevent these from going between the rails. The width of the machine can also be made rather less, as in the Rigg machine some extra width is required to allow room for the extra projection of the bevel-wheel, which gears into the rack on the disc. The question of the width is one of very great importance where the tendency of the roof is to be soft.

The general effect of the higher speed of the Rigg machine seems, in practice, to be to make it more suitable for a coal-holing, and the Gillott cutter, on the other hand, is generally more successful where the holing is fireclay or daugh. The direction of the motion of the disc seems to matter little in practice.

The holing of the virtuewell coal-seam at Glenclelland colliery, for which the cutter was designed, is daugh and fireclay, varying in thickness, but generally easily cut. As thought most suitable for this holing, the disc of the machine was designed to run at a comparatively slow speed —about 25 revolutions per minute—and there are 20 picks on the circumference, fixed somewhat similarly to those in the Gillott machine. The disc, on the other hand, is supported in its place similarly to the Rigg machine, and cuts level with the pavement. The cut is 3 feet in depth and about 3½ inches in height. It will be seen from Figs. 1 and 2, Plate VI., that the speed of the motor is reduced by gearing, there being in all four shafts, including the armature-shaft of the motor. The disc is turned by means of a bevel-wheel, which gears into a rack on the disc inside the circumference.

For oiling the bearing of the disc, there is a large cup in the cast-iron bracket supporting the disc; this cup is covered by a sheet-iron lid

(which is screwed on), and is filled once in each shift with waste soaked in oil. The brass under the bracket near the frame, which keeps the disc horizontal, cannot in practice be lubricated at all.

The machine is drawn forward by means of a flexible wire rope, which passes round a drum in the framework of the machine, round a pulley fixed to a prop some distance in front, and back to the machine, where it is fixed to the frame. The drum is turned by means of an eccentric on one of the shafts, which gives a reciprocating motion to a pawl; this catches the teeth of a ratchet wheel, keyed on a shaft which is geared to the shaft whereon the drum is fixed. By means of a slot, the feed may be varied, the pawl taking one to three teeth for each motion as desired.

On the first machine, there was a friction-arrangement in connexion with the drum, so that it might slip with any extra strain, but there was considerable trouble with slipping when slipping was not wanted, and the arrangement was abandoned.

The switch for reversing the motor has two sets of resistances and three steps, so that the current may be gradually turned off or on. There is also a small auxiliary rapid cut-off switch, which has been found convenient for use throughout the shift, and which is cheap and easily repaired.

Working of the Cutter.—Four men are required to work the machine. One man attends to the switch, etc., and sees that the disc is cutting rightly; he also hands the rails and sleepers over the machine, and puts up gibs along with another man who also shovels out the cuttings. The other two men lay the rails and put up timber.

A great part of the work consists in handling the rails, and on the expedition with which this is done greatly depends the length of wall that can be cut.

The machine has regularly cut 420 feet in length, 3 feet deep, in a shift of 10 hours. When cutting in the ordinary holing it easily cuts 3 feet in 1½ minutes, and can do so continuously when it is kept in rails. From the diagrams of the ampèreograph it can be seen that there are considerable stoppages. This instrument was not put into circuit until the length of wall was reduced, owing to a number of places being marched, but the following are examples of what is shown for a cut of 210 feet:—

	7th March.		9th March.
Machine working ...	10 minutes	...	25 minutes
„ standing ...	15 „	...	10 „
„ working ...	10 „	...	33 „
„ standing ...	18 „	...	15 „
„ working ...	15 „	...	20 „
„ standing ...	3 „	...	5 „
„ working ...	20 „	...	30 „
„ standing ...	10 „	...	5 „
„ working ...	40 „	...	13 „
„ standing ...	10 „	...	40 „
„ working ...	30 „	...	70 „
„ standing ...	25 „	...	8 „
„ working ...	45 „	...	5 „
„ standing ...	10 „		
„ working ...	20 „	Total	279 „
Tptal ...	281 „		

Sixty to seventy minutes at a time is about the longest interval during which the machine has been shown by the ampèreograph to be in continuous operation. Owing to the sensitiveness of the ampèreograph, there is too much vibration in the movement of the pen for the markings to be reproduced satisfactorily.

The experience in working the machine has called attention to certain practical points, some of which may be noted here:—

As has been stated, the weight of the second coal-cutter is about 32 cwts., and of the one which has been working somewhat more, considerably heavier in both cases than any of the machines worked by compressed-air. The extra weight is an advantage in giving steadiness while the cut is being made. There is less vibration than with compressed-air machines, partly on account of this extra weight, and partly because there is no reciprocating motion.

The extra weight, on the other hand, has several disadvantages. Heavier rails are required, which require more labour for shifting and laying. When anything goes wrong, the machine is awkward to handle, and there is additional expense in shifting into the new cut at the beginning of each shift. The first machine supplied takes about one to one and a half hours of three men to bring it forward to its new position, and this adds something like one-ninth to the cost of cutting each wall. There is an arrangement in the new machine, however, which it is expected will save much of this time. It will be seen from Fig. 1 that there are notches in each end of the frame, for the insertion of rails on which the machine should be easily moved. As these rails must be clear of the picks on the disc, and sufficiently far from the cross-bars on the ends of

the frame to leave room for a lifting screw to work, the framework is, of necessity, made rather longer than otherwise necessary. This is some disadvantage.

Some difficulty was found at first, when the disc got jammed in the cut, in getting the machine clear. In such circumstances, the pawl being caught firmly in the ratchet-wheel, any motion of the motor increased the difficulty, and the motor could not be reversed without some loss of time. In order to get over this, there is a swivel with screw-attachment for the pulley, round which the drawing-rope turns. By turning this swivel and screw the rope may be slackened, to enable the machine to be moved a few inches out of the cut and the disc to be relieved.

To keep the rails firmly in position, and to resist the extra strains which at times tend to thrust the machine from the face, every second or third sleeper has projecting from the end farthest from the face, a piece of iron turned into a horse-shoe shape, into which a prop may be inserted, and fixed firmly between the roof and the pavement.

The wall at first in operation was at an inclination of about 1 in 8, and in coming down the machine required to be held back. At first this was done by means of a wire rope and drum, but it was found better to use a goose-arrangement to run on the rails in place of the ordinary wheels. This goose is made of malleable iron, and is all in one piece, the two ends being joined together by a bar which has two holes, into which are slipped two pins, which pass through holes at the end of the framework of the machine. The arrangement works satisfactorily.

Tests.—The writer regrets that the tests which he is able to submit are not so satisfactory as could be wished. When the testing instruments were available, the length of the wall was only about 210 feet, and on the occasion when the tests were made, the butterfly-valve, working in connexion with the governor of the engine, was out of order. Every care was taken to keep the voltage as steady as possible, by carefully watching the voltmeter and regulating the speed by hand, but errors may have crept in where simultaneous readings were necessary above and below-ground. The figures are, however, given as recorded, and further tests may be made before the next meeting.

It is readily seen from Table I. that the plant is not working to full advantage. The engine is large enough for five or six times the work that is being done, the dynamo shows a greater percentage of loss than would be the case if it were working with a heavier load, and the efficiency would be very much greater if there were more cutters at work. The resistance of the shunt of the dynamo is 180 ohms, and the current that

is passing through it is about 2 ampères. This is in addition to the amount of current shown by the ampèremeter. There appears to be a

TABLE I.—EXPERIMENTS.

	Revolutions per Minute.	Indicated Horse-power.	Volts.	Ampères.	Electric Horse-power.
Engine running light	50	4·1	—	—	—
Engine with dynamo and switches	50	10·5	—	—	—
Engine with full load	50	14·2	—	—	—
Dynamo	750	—	300	—	—
Motor and cutter	800	—	275	15	5·5
Loss in cables	—	—	25	—	0·5
Loss in shunt of dynamo	—	—	—	2	0·8
Loss in leakage	—	—	—	5	2·0

leak somewhere of about 5 ampères (from the indications of the ampèreograph), and it is probably due to the cables in the shaft being only temporary and not sufficiently insulated. The motor is not using nearly as much power as was expected.

Results of Working.—In connexion with an installation of this kind, probably the first question that will be asked is, what is the advantage of the use of machinery in place of hand-labour? The writer regrets that he is unable to give a very decided answer. The machine has hitherto not been working to the best advantage for various reasons.

The section of the virtuewell coal-seam at Glenclelland colliery, taken from one of the machine walls, is as follows :—

	Ft.	In.	Ft.	In.
Roof—				
Roughband ironstone			0	8
Fakey blaes			1	6
Brushing—				
Fakes			3	6
Working—				
Black shaley blaes	0	5		
Good gas shale	0	4		
Shale and blaes	0	9		
Virtuewell coal	2	6		
Daugh holing...	0	3		
			4	3
Hard pavement—				

The coal itself has a bad parting from the roof, and it has been found best to take the working to the height shown, the 14 inches of shaley blaes being turned over into the stowing, after the 4 inches of good shale has been picked out to be sent to the surface. The coal has to be blasted.

Owing to the extra thickness of brushing at this portion of the field, the walls were driven 60 feet in length, instead of 45 feet as usual in

other parts of the seam, to save expense. It was thought that two men might be able to fill the coal from 60 feet, but this was found on trial to be too large a quantity, and in consequence it was found practically impossible for the machine to work every shift. More than two men could not work at each wall owing to the large turnover of rubbish. This irregularity added to the expense of working. One reason why the filling was more difficult to manage than anticipated was that the blaes fell over in large pieces, which were difficult to handle. When the coal is worked by hand the holing is not so deep, and the pieces of blaes are smaller and more easily dealt with.

It is expected when the machine is in full operation in the new wall, which is in a direction at right-angles to the old one and level, that less blasting will be required to bring down the coal, and that the cost of filling will be considerably reduced.

Meantime the comparison between the cost of working by hand and by machine may be taken as follows :—

	Per Ton. s.	d.
Cutting and filling by hand... ... ... 1s. 8d. to	1	10
Cutting by machine, including cutting-out at each end of the wall	0	3·5
Filling, which has varied from 1s. 1d. to 1s. 3d. per ton...	1	3·0
Fuel (estimated)	0	0·5
Upkeep and attendance on surface, sharpening picks, etc.	0	1·5
Interest and depreciation	0	1·5
Total	1	10

This statement shows no saving, but there are other points which must be considered, besides the actual cost per ton at the face. There is less dross and larger coal produced from the machine walls. There is a much larger output from the same faces, and therefore a less cost of upkeep for roads. There being a greater concentration of output, the haulage can be more economically managed. On the other hand, the wall advances so quickly that the driving of slope-roads will be more costly than in ordinary circumstances, as they must be made more expeditiously than is usually the case.

Various savings can be made in the items named when there are more cutters at work, and the writer anticipates that while there may not be a great saving over hand-labour at Glenclelland colliery, yet there will be some clear advantage in the use of machines. It seems to be very much, as often stated, that the use of machines is only advantageous if the

Patented ... at Glenclelland Colliery."

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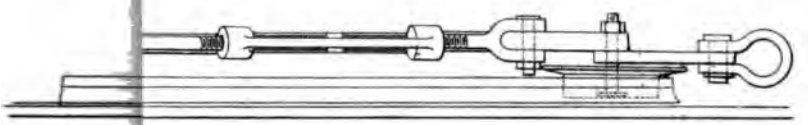

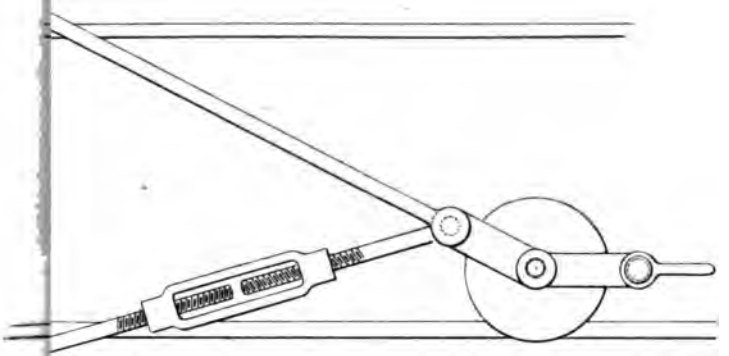

Mine

VOL. XVI. PLATE IV.

circumstances are favourable, and that there is no great saving in ordinary cases when wages are low. The essentials are a clean field and a strong roof, and the harder the holing the greater will be the saving. As regards the use of electricity instead of compressed-air, there seems little doubt that the former method of transmitting power must supersede the latter, except possibly where there is fire-damp present, or where the workings are wet. There is some danger from the sparking of the brushes in an explosive mixture, though this difficulty will no doubt be got rid of in the future. Meantime caution must be exercised. With water falling it might be difficult to keep the resistance of the armature right, and any colliery owners who have wet workings should consider this defect before adopting electricity.

The only advantages of electricity that need be named here are the greater efficiency, the convenience of erecting wires instead of laying heavy pipes, and the facility with which the results of cutting may be observed. There is also the feature that the power given out by the dynamo is in proportion to the power required at the motor, and may rise, temporarily, without damage much above the average, when for any reason extra power is required.

From experience of the machinery, breakdowns are no more likely to occur with electric than with compressed-air machines, and the parts that require renewal and repair are the parts which are the same in machines driven by either power.

There is very little difference in the first cost between plants worked by compressed-air and by electricity. The cost of upkeep should be much the same. For very thin seams, lighter machines should be made. This can be done by giving the motor a higher speed, and by working with less power. It is by no means clear that a lighter machine cutting more slowly will not cut nearly as much as the heavier machine, as it might be possible to keep it cutting more regularly with fewer stoppages. An important point about all machines is the regularity of feed, jerks should be avoided, and the aim should be to keep the cutter travelling forward as steadily as possible.

The following paper, by Mr. Dugald Baird, on "Pumping-Engine Velocity Diagrams," was read by the Secretary :—

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The following paper, by Mr. Dugald Baird, on "Pumping-Engine Velocity Diagrams," was read by the Secretary :—

PUMPING-ENGINE VELOCITY DIAGRAMS.

By DUGALD BAIRD.

In almost every instance when a paper on pumping is read before the Institute, the speed of the ram of the pump is one of the main things discussed.

Hitherto, so far as the writer is aware, the speed of the ram has always been spoken of as so many feet per minute, which was found by multiplying the length of the stroke by the number of single strokes per minute. Now, it is obvious that this style of calculation does not give the actual speed of the ram, but only its average speed spread over one minute of time. If an engine with a 10 feet stroke is found to be going at five strokes per minute, then the speed of the ram is said to be 100 feet per minute. If this engine is a direct-acting one of the Cornish, bull, or other similar type it will be found to make a distinct pause of possibly two seconds at each end of the stroke. This at once reduces the actual time that the ram is in motion from twelve to eight seconds per double stroke. The average velocity of the ram during one double stroke would then be not 100 feet per minute, but 150 feet per minute. But we next find that the upstroke in the pit (if the engine be single-acting) is made much more rapidly than the down-stroke, and so the average velocity of the one stroke is very different from that of the other. We know also that the first part of each stroke and the last part of each stroke are much less in velocity than the central part of it, because the ram has to start from rest at one end and come to rest at the other, so that one might guess the speed in the centre of the stroke in this case stated at perhaps 300 feet per minute; but this would vary very much in different types of pumping-engines.

In geared bell-crank engines, it could be approximately calculated if the engine had a large fly-wheel and were going rapidly, thus giving almost uniform motion, but with other types of engines it could not be calculated, and would, therefore, if actually wanted, require to be guessed. Now, this is a very unsatisfactory process, and to obviate this a machine has been designed by the writer, and although (as presently constructed) it is not so perfect as might be, it is claimed that it is capable, if properly made, of giving the desired information, and that it is correct in principle.

This machine, as shown on Fig. 1, Plate VII., was originally constructed to be hand-driven, and the diagrams (Figs. 2 to 6, Plate VII.) have all been taken while it was so driven; clock-work has now been applied to it, and the diagrams which may be taken by it in future should be more accurate. The machine consists of a base *a*; a vertical drum or cylinder *b*; and two vertical guide-bars *c*, upon which moves a heavy pencil-holder *d*. There are gear-wheels *e*, *f*, and *g*; and *h* is the crank by which it is driven. The crank *h* makes 30 revolutions for one of the drum *b*.

While the drum is in motion, the pencil is attached by means of small brass wire or a cord to some part of the engine which will give a stroke to correspond to the stroke of the engine, and of such a length as will suit the machine. It will be seen that if a sheet of paper be fixed on the drum and the pencil made to act when the engine is standing while the drum is revolving, a horizontal line will be drawn on the paper, while if the engine be going and the drum stationary a vertical line will be drawn on the paper; but if they be both moving at one time an irregular figure will be formed according to the action of the engine, as shown in Figs. 2, 3, 4, 5, and 6, Plate VII. The vertical lines are

VELOCITY TABLE.

No. of Diagrams.	Type of Pumping-Engine.	Speed per Minute in Strokes.	Length of Stroke.	Average Velocity of Two Single Strokes per Minute.	Average Velocity of Up-stroke per Minute.	Average Velocity of Down-stroke per Minute.	Greatest Velocity of Up-stroke per Minute.	Greatest Velocity of Down-stroke per Minute.
			Feet.	Feet.	Feet.	Feet.	Feet.	Feet.
2	Compound differential...	4	7	56	105	95	140	135
3	Compound differential...	4	10	80	237	175	315	285
4	Single cylinder bull ...	2¼	11½	63	157	119	200	150
5	Compound differential...	3¾	11½	86	138	144	172	200
6	Compound beam ...	2¼	12½	56½	125	161	162	202
7	Bell crank	6	4	48	48	48	75·4	75·4

drawn at intervals of one second, and the horizontal lines at intervals of one foot, so that the velocity can be easily measured by taking these intervals as the scale of the diagrams.

The table shows that the greatest velocity in the pumping-engines tested varied from twice to about four times the average velocity of two single strokes.

None of these engines were hard driven, the greatest average velocity being 86 feet per minute, and the greatest actual velocity being 315 feet per minute. Most pumping-engines can be driven without fear of break-down to an average velocity of 100 feet per minute, and some, especially

those of the differential-gear type, can make 130 feet per minute with ease. If a velocity diagram were taken at either of these speeds the greatest velocity would probably be 350 feet per minute, the balance of the gain being got from a short pause at the terminals of the stroke.

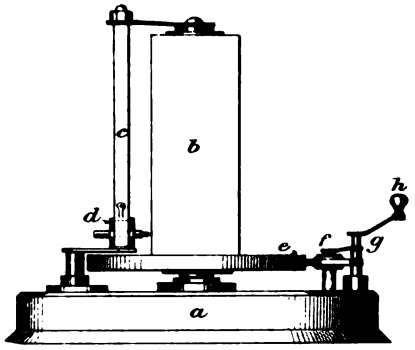
Since these diagrams were taken, it has come to the writer's knowledge that velocity diagrams were taken some years ago, but he has not seen any description of the machine by which they were taken. From these diagrams it appeared that a Cornish pumping-engine made the great velocity of 606 feet per minute during the steam stroke, and of 228 feet during the water stroke.

Most people will admit that other things being equal the best pumping-engine is the one that can pump a certain growth of water at the lowest velocity of ram. When tried by this test the Cornish pumping-engine will be found to be defective. One fears to say a single disparaging word against a style of engine which has done so much excellent work in the past, and which is still doing good work ; but the writer is of opinion that the Cornish pumping-engine is gradually becoming a thing of the past, and the excessive speed at which it makes its steam stroke is very much against it. The idea aimed at and attained in the Cornish pumping-engine was to get a high initial pressure and a quick cut-off, thus getting a large expansion in a single cylinder engine, but now a larger degree of expansion is obtained by using a compound engine at less velocity. The initial pressure in the Cornish pumping-engine is usually more than twice the average pressure, whereas that of the modern compound pumping-engine is about $1\frac{1}{4}$ times the average pressure in the high-pressure cylinder.

It naturally follows that the liability to break-down is less in the modern engine, while the economy does not suffer much, if any, compared with the Cornish pumping-engine. Short lifts, and many of them, is one of the leading features of the old Cornish pumping-engine, and this with the heavy aprons and glands required at each lift, and everything requiring to be made heavy and strong so as to withstand the rapid steam stroke of this engine gave a ponderous mass which often exceeded by twice or thrice the weight of water lifted, the surplus weight being counterbalanced. This mass requiring to be started and stopped at each single stroke provided the means whereby a high initial pressure and quick cut-off could be got, and without which no such expansion as has been got would have been possible.

It is but fair here to state that even at the present time when we find a Cornish engine with all its fittings suitably applied and proportioned it gives out duty which has not yet been surpassed, although equalled, by the most modern type of compound engine.

FIG. 1.



ELEVATION.

SCALE 8 INCHES TO 1 INCH.

FIG. 2.

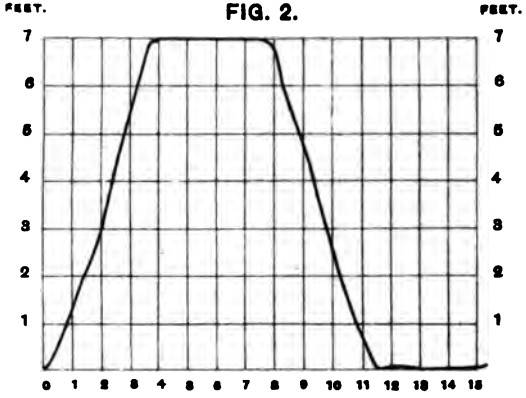


FIG. 3.

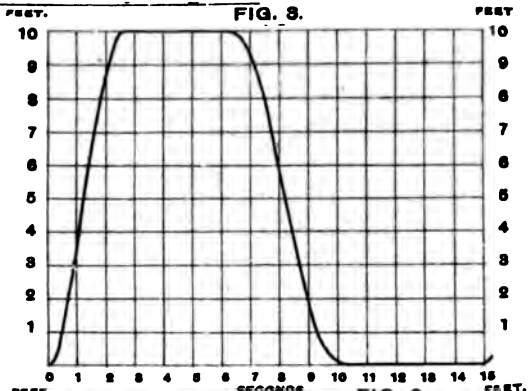


FIG. 4.

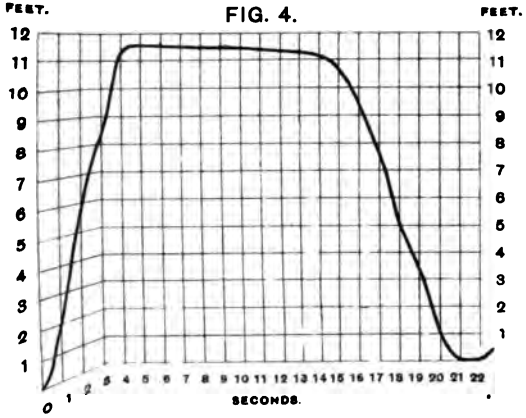


FIG. 5.

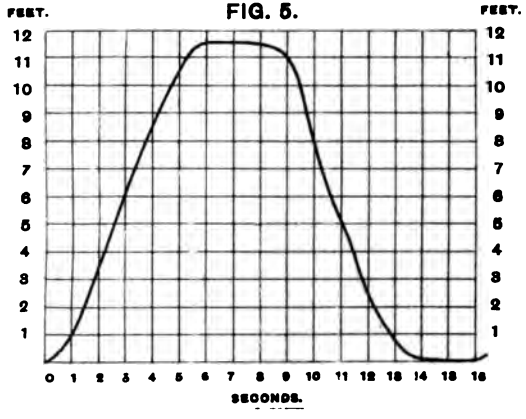
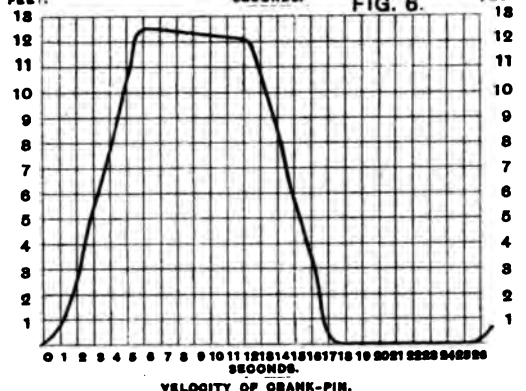
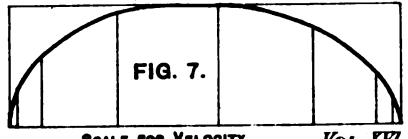


FIG. 6.



VELOCITY OF CRANK-PIN.

FIG. 7.



SCALE FOR VELOCITY.
2 FEET PER SECOND TO 1 INCH.

Vol. XVI Plate V.

The diagram (Fig. 7, Plate VII.) of a geared bell-crank pumping-engine is made from calculations and in this case the motion of the pumping crank-pin is taken as being uniform. The pumping-engine (with a stroke of 4 feet and connecting-rod 13 feet long) was running at a speed of six revolutions per minute. The maximum velocity of the ram in this case was only 1·57 times the average velocity or 157 feet per per minute at an average velocity of 100 feet per minute. This speed looks very favourable for the geared bell-crank engine, but the serious objection to this engine is the want of a proper pause at the termination of the stroke. It no sooner reaches the end of one stroke than it moves away for the next stroke, and the valve lids do not get time to seat themselves, with the double defect of slip and shock. With bucket-pumps this is not felt to the same extent as with ram-pumps, owing no doubt to the usual short vertical distance for which a bucket is employed and the favourable situation of the bucket-lids.

By means of these velocity diagrams we are enabled to see at a glance the actual speed of the ram at all points of the stroke and to note the duration of the pause, a result which is both interesting and useful.

The machine shown on Fig. 1 is far from being perfect; it was made at the colliery workshops and is somewhat clumsy. The writer would suggest that the Council should get a similar but smaller instrument made by a maker of scientific instruments, and give it to the pumping committee of this Institute; the instrument to be of such a size as could be comfortably carried about from place to place, so that the various pumping-engines about to be indicated for pressures should also be indicated for velocities.

The writer is strongly of opinion that the pumping-engine that can do a certain amount of work at the slowest, and nearest to a uniform velocity, and can give a good pause at the end of the stroke with no violent pressures, is superior to one that does its work by leaps and bounds, and a machine as above described is highly necessary if a good and fair comparison of different types of pumping-engines is to be made.

DISCUSSION ON MR. A. M. GRANT'S "REMARKS ON WINDING-ENGINES."*

Mr. JAMES FREW (Dunaskin) asked if Mr. Grant could give any details of the Newbattle winding-engine?

The PRESIDENT said that a portion of Mr. Grant's paper had not been printed owing to revision by the Publication Committee of The

* *Trans. Fed. Inst.*, vol. viii., page 390.

Federated Institution of Mining Engineers. The specification of the winding-engine at Newbattle colliery was for some reason omitted. Mr. Grant would be prepared to answer on any other part of his paper at the next meeting.

Mr. FREW thought that Mr. Grant had some objection to piston-valves as regarded winding-engines. Could he give them any reason for that objection?

DISCUSSION ON MR. J. G. BARCLAY'S PAPERS ON "THE BARCLAY CROWN-VALVE EXHAUST-GEAR,"* AND "AN OVERHANGING BEAM PUMPING-ENGINE."†

Mr. JAMES FREW (Dunaskin) said Mr. Barclay claimed that by having the air-pump and condenser placed as shown in Plate XXIII., the stroke of the air-pump was reduced one-half, say, 3 feet, while the area of the air-pump was doubled. Mr. Barclay also claimed that there was a more perfect vacuum with the 3 feet stroke than with the 6 feet stroke, the cubical contents of the discharge being the same in either case. He (Mr. Frew) concurred with Mr. Barclay's claim, as, with the long stroke, no other power existed than the back pressure between the cylinder piston and the water-level in the condenser to cause the water to leave the condenser and rise the height necessary to follow the air-pump bucket for a full discharge. In practice, however, it was usual to place the condenser so that a minimum rise of water-level in the air-pump barrel was attained above the water-level in the condenser, when the air-pump bucket had reached the top of the stroke; and in many cases the condenser was set higher than the air-pump discharge, so that the air-pump was filled by gravity alone. Many jet-condensers without foot-valves were at work, and work efficiently. In such cases the cold water-cistern surrounds the condenser and air-pump, thus combining surface and jet-condensation in one apparatus, the whole question being one of arrangement.

The discussion was adjourned.

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* *Trans. Fed. Inst.*, vol. viii., page 396. † *Ibid.*, page 397.

MIDLAND INSTITUTE OF MINING, CIVIL, AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD AT THE INSTITUTE ROOMS, BARNSLEY, FEBRUARY 9TH, 1895.

MR. H. B. NASH, VICE-PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and confirmed.

The following gentlemen were elected, having been previously nominated:—

MEMBERS—

Mr. DAVID BANNISTER, Mechanical Engineer, Shaw Cross, near Dewsbury.

Mr. AUGUSTUS BEVER, Mechanical Engineer, Dewsbury Foundry, Dewsbury.

Mr. WILLIAM THOMAS DEPLEDGE, Timber Merchant, The Lodge. Stanley, Wakefield.

Mr. DAVID ELLIOTT, Colliery Under-Manager, Houghton Main Colliery, Barnsley.

Mr. WM. FIRTH, Iron Merchant, Water Lane, Leeds.

Mr. JONATHAN ISHERWOOD, Colliery Manager, Hoydroyd Colliery, Barnsley.

Mr. JAMES REEVES, Jun., Assistant Colliery Under-Manager, High Carr House, Alvecote, Tamworth.

Mr. ARTHUR SHUTTLEWORTH, Colliery Under-Manager, Park Terrace, Worsbro' Park, near Barnsley.

Mr. ARTHUR SIMPKIN, Assistant Colliery Manager, South Kirkby, near Wakefield.

Mr. JAMES TAIT, Colliery Under-Manager, South Kirkby Colliery, near Wakefield.

MIDLAND INSTITUTE OF MINING, CIVIL, AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD AT THE QUEEN'S HOTEL, LEEDS, APRIL 6TH, 1895.

MR. J. NEVIN, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and confirmed.

The following gentlemen were elected, having been previously nominated:—

MEMBERS—

Mr. FRED COULTAS, Colliery Manager, Deepcar, near Sheffield.

Mr. ROBERT WATKIN, Colliery Under-Manager, Cortonwood Colliery, Barnsley.

Mr. THOS. FOSTER, Colliery Under-Manager, Oaks Colliery House, near Barnsley.

Mr. WALTER NEWTON DREW, Mining Student, Rockingham Colliery, near Barnsley.

Mr. GEORGE CHARLES KENYON, Colliery Under-Manager, Sandygate, Wath, near Rotherham.

Mr. SAMUEL DAVIES, Deputy, 3, Park Road View, Worsbro' Bridge, Barnsley.

Mr. H. WORMALD read the following paper on "Designs for Pump-valves":—

DESIGNS FOR PUMP-VALVES.

By H. WORMALD.

The pump-valves referred to in this paper are to a great extent intended for use with clear water, although some of them are also suitable for dealing with dirty water.

Pump-valves may be divided into three classes, viz. :—

(*a*) Valves suitable for low lifts,

(*b*) Valves suitable for medium lifts, and

(*c*) Valves suitable for heavy lifts.

The arrows in the various illustrations show the direction in which the water flows through each particular valve. The rubber rings shown around the central spindles are intended to be compressed during the opening of the valve, the opening being thereby regulated according to the volume of water passing through ; and to some extent the amount of wear and tear in any particular valve will vary according to its work.

(*a*) *Low-lift Valves.*

Low-lift valves are suitable for lifts, say up to 200 feet high.

The valve shown in Figs. 1 and 2 (Plate VIII.) consists of an ordinary indiarubber disc *A*, working on a grid or grating *B*. This valve can be made almost of any size to suit the quantity of water pumped, but care must be taken in constructing the grid or seat. Each of the holes in the grid should be of a suitable size, so that the indiarubber disc will not be squeezed through under working conditions. The guard *C* should be constructed with holes as shown in Fig. 2, in order that the water-pressure can get behind, and close the valve quickly at the end of the stroke.

Figs. 3 and 4 show the pyramid valve, consisting of two circular indiarubber rings *A A*, fixed one above the other, and held down by the guards *B*. In working, the outer edges of each of the indiarubber rings rise against the guards, and so allow the water to pass through the openings in the grating or seat *C*. These openings should be of the same design as those shown in Figs. 1 and 2. This valve has an ample area for the passage of the water through it.

Figs. 5 and 6 show the hatband valve, consisting of three circular indiarubber rings *A A A*, placed one above the other. These valves are secured to the valve-seat *B* by means of wrought-iron hoops *C*. The indiarubber rings only open on the top side and allow the water to pass through the openings in the grating or seat. This valve affords a large area in the seat for the passage of the water, and more especially as there are no ribs or bosses to impede its progress.

Figs. 7 and 8 show the ordinary double-flap or butterfly valve, which is extensively used for pumping purposes. In this case, the valves *A A*, are made of leather strengthened and weighted by wrought-iron plates *B B*, so as to facilitate their closing under working conditions. The valves shown in the figures are hinged with leather, open under the crossbar *C*, and are prevented from opening too far by the stop *D*.

Fig. 9 shows the same class of butterfly valve as Figs. 7 and 8, but in order to make the working conditions more clear, the bucket or delivery-valve and the suction-valve are shown combined. The outer coating *E* in the bucket is made of leather or guttapercha held in position by the wrought-iron ring *F*. The outer coating *E* is easily replaced when necessary by removing the crossbar *G*.

(*b*) *Medium-lift Valves.*

Medium-lift valves are most suitable for lifts, say up to about 400 feet high.

Fig. 10 shows the ordinary single-flap or clack-valve and bucket combined. The lower valve or clack-seat *A* is fixed at the bottom of the working-barrel, and the valve *B* swings on the pin *C*, and is prevented from opening too far by the stop *D*. The bucket *E* is of the same construction as the suction-valve, and is turned on the outer edge so as to fit the working-barrel *F*. The valves shown in the figures are made of brass, and are much too large and too expensive for ordinary purposes. The usual plan of making a bucket for an ordinary house-pump, is to take a piece of wood and turn it in the lathe to suit the size of barrel, bore a hole in it for the passage of the water, and nail a piece of leather on the top of it to act as the lid or valve.

Figs. 11 and 12 show a type of an ordinary leather disc flat-beat valve, with a large water-way, as shown by the openings in Fig. 12. This valve is not suitable for use when pumping dirty water, because as the valve in opening rides bodily up the central spindle *A*, the sediment contained in the dirty water would lodge at the root of the central spindle, and prevent the valve from closing.

Figs. 13 and 14 show the ordinary flat metallic double-beat valve. The valve-seat *A* is made of cast-iron, principally with the object of keeping down the cost. The beats *B* are of gun-metal fixed in the seat, and the valve *E* is of brass. The water is free to pass away from the inner beats *C*, as well as the outer beats *D*. This valve is extensively used at waterworks, and gives excellent results.

Figs. 15 and 16 show a valve constructed on the same lines as Figs. 13 and 14, save that it is of much cheaper construction, but is equally efficient. The beats *A A* can be made of leather, guttapercha, brass, wood, or anything suitable to the conditions under which it may have to wo .

(*c*) *Heavy-lift Valves.*

The following valves are suitable for heavy lifts above 400 feet high :—

Figs. 17 and 18 show an ordinary single-beat mushroom valve, having a spindle *A* guided in the cover of the valve-box, and a central spindle *B* on the underside of the valve guided in the valve-seat *C*.

Figs. 19 and 20 show an ordinary single-beat wing valve, having a central spindle *A* on the top of the valve guided in the cover of the valve-box, and four wings or ribs *B* on the underside of the valve, which act as guides in the inner circle of the valve-seat *C*.

The valves and seats shown in Figs. 17, 18, 19, and 20 are usually made of brass, and are adapted for almost any pressure. As they are only single-beat valves, there is often a good deal of noise when working, especially when the lift is heavy and the speed fast. These valves are, however, more extensively used than any other class for general pumping purposes.

Figs. 21 and 22 show a type of double-beat valve, which is suitable for either high or medium lifts. The beats are arranged so that they give a free passage to the water, and the tendency to bring the valve back to its seat with a sharp click is considerably less than that of the ordinary single-beat valve, because the area of valve exposed to the water pressure is considerably less. The beats *A A* can be made of leather, guttapercha, gun-metal, white metal, or other substance which may be suitable for the purpose. The valves rise bodily on the central spindle *B*, which acts as a guide, and the main body of the valve rides on the wings *C C* formed in the valve-seating.

Figs. 23 and 24 show another type of double-beat valve, similar in construction to the last described, the difference being that the beats *A*

are arranged at an angle, so as to allow the water—as indicated by the arrows—to continue nearly in a straight line on its course through the valves, thereby diminishing the loss of power absorbed in friction.

It will be noticed that the author has not entered minutely into the material of which any particular valve should be made. This selection depends upon the chemical properties of the water to be pumped and upon the predilections of those ordering the pumps.

The progress made during late years in the design of pump-valves is amply proved by the fact that it is common practice to raise in one lift at the present day what would have been raised in five or six lifts thirty or forty years ago.

Mr. A. LUPTON (Leeds) asked why, apart from economical reasons, engineers were so fond of using gun-metal and brass for valves. Some years ago he had to design valves to stand a pressure of 20 to 30 tons per square inch. The valves were made of steel, with steel seats or taps, and were competent to withstand that enormous pressure. If in the case of valves standing that enormous pressure steel was used, why did engineers generally use gun-metal? Again, had Mr. Wormald considered the relative advantages of a number of small valves—six, seven, or eight valves in one bedplate, as against one large valve in the case of a large ram, say 20 inches in diameter or larger? He did not see why the valve shown in Fig. 21 (Plate VIII.) should be more suitable for a high lift than the valve shown in Fig. 13. With regard to guttapercha seats he had found them suitable for heads of 800 or 900 feet. Guttapercha was found to stand severe usage for a long time very economically.

Mr. WORMALD replied that gun-metal was perhaps the best material for the beats of valves for the pressures dealt with in his paper, but it was not suitable for pressures of 20 or 30 tons per square inch—for such pressure the best hardened steel should be employed. A pressure of 20 or 30 tons per square inch would crush a beat made of ordinary gun-metal. With greater pressures, harder metal and larger beats were needed. A series of valves should be used in the case of large pumping machinery, which it might be desirable to run at a slow speed. In an engine equal to 8 or 10 strokes per minute, if the water-supply was only equal to 1 stroke per minute, and they wanted to keep the engine working at that slow speed, there was no objection to one lifting valve out of the series of seven, being used singly, thereby saving the

To illustrate Mr H. Wormald's Designs for Pump-valves.

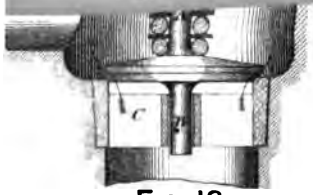


Fig. 18

References

WROUGHT IRON

CAST IRON

INDIA RUBBER

BRASS

Fig. 19.

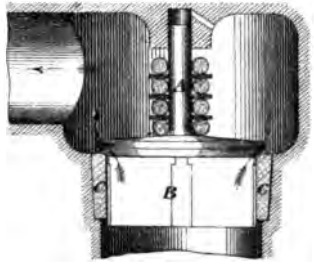


Fig. 20.

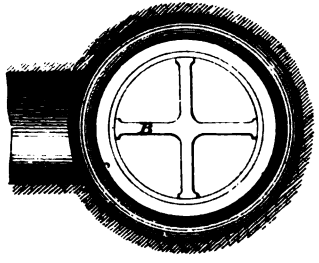


Fig. 21

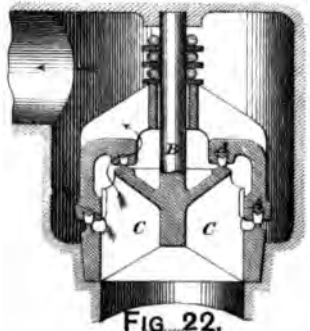


Fig. 22.

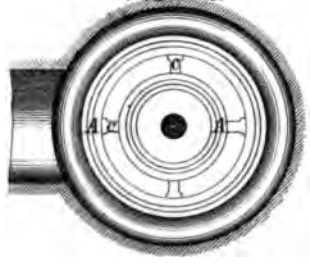


Fig. 23.

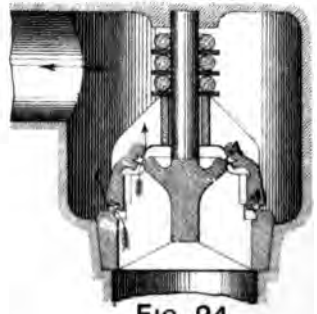
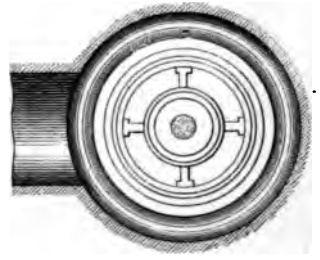


Fig. 24.



Midland Institute of Mining, Civil & Mechanical Engineers, Transactions 1894-5.

And.ᵛ Reid & Comp.ʸ L.ᵗᵈ Newcastle on Tyne

other valves from wear and tear. The valve shown in Fig. 3 (Plate VIII.) was used in the York waterworks, but he gave the preference for heavy work to the valve shown in Fig. 21.

Mr. T. W. H. MITCHELL said in several of the valves the guttapercha or leather was attached to the lift and not on the seat. Was there any special reason for this arrangement? Many brass valves were working satisfactorily with guttapercha fixed into the seat, working at a head of 900 feet. Was there any practical advantage in having the joint-making substance placed in the lift instead of in the seat?

Mr. WORMALD, in reply, said one advantage was that the valve could be easily taken out, whereas the seat of the valve shown in Fig. 13 was more difficult to take out and repair. If the guttapercha wore away the whole valve must be taken out and turned up to suit the valve-seat, and the valve would be worn gradually by having to repair it in that way; whereas if the seat of the valve shown in Fig. 21 wore away it could be replaced, and the whole valve remained as good as when it was put in.

Mr. A. LUPTON moved that a vote of thanks be accorded to Mr. Wormald for his paper.

Mr. E. W. THIRKELL seconded the motion, which was carried unanimously.

Mr. R. ROUTLEDGE read the following paper on "The Garforth Collieries, with special reference to the Failures of the Tubbing and Inundations which occurred in 1872 and 1883":—

THE GARFORTH COLLIERIES, WITH SPECIAL REFERENCE TO THE FAILURES OF THE TUBBING AND INUNDATIONS WHICH OCCURRED IN 1872 AND 1883.

By ROBERT ROUTLEDGE.

The Garforth collieries are on the Parlington estate, situate 7 miles east of Leeds, and connected by a private line to the North Eastern Railway. The coal-bearing strata on the estate extend over an area of from 5,000 to 6,000 acres. The seams of coal which crop out in the northern portion of the property dip at a considerable angle towards the south. In former years, and long before the existence of railways, many shallow pits were sunk, from which the coal was worked to supply local requirements. The coal-seams which have already been proved comprise the following:—

	Ft.	Ins.
Firthfield or brown metal	3	6
Middleton little or hard band...	5	2
Middleton main or bright coal	4	6
Barcelona or blocking coal	2	0
Beeston bed	5	0
Total thickness of coal	20	2

It is expected that the black bed and better bed coal-seams will be found below the Beeston seam, as they have been proved at the Micklefield colliery, the former at 210 feet and the latter at 270 feet below the Beeston coal-seam.

Immediately below the bright coal-seam is found the water-bearing stratum, which consists of a thick bed of sandstone, much broken up by gullets, and full of water. This bed is about 120 feet thick, and is believed to thicken towards the south, the Beeston coal-seam being 180 feet below.

A considerable area of the Firthfield, hard band, and bright coal-seams have been worked by the old pits already referred to, but as the workings extended to the dip, the appliances of the period were unable to cope with the large quantities of water met with, and they were then abandoned. Up to this time the mines were drained by means of a water-level adit, which was driven from the bottom of the Firthpit field

for a distance of 2 miles, where it discharged into the Kippax beck, a feeder of from 1,000 to 1,500 gallons of water per minute. This water-level is in use to the present day, and keeps the sandstone feeder a little below the top of the tubbing in the Sisters' shaft.

In order to prevent the dip workings from being flooded by infiltration from the outcrop a water-level drift, running from east to west, was also constructed at great expense; it carries off all surface-water, and is still in fairly good order.

Two shafts were sunk, the Isabella shaft in 1833, and the Sisters' shaft in 1843, through the water-bearing stratum to the Beeston bed, and the feeders were tubbed off. The Sisters' shaft is 495 feet deep, and is tubbed for a length of 255 feet through the sandstone, commencing at a depth of 120 feet from the surface, and terminating at a height of 120 feet above the Beeston coal-seam. It is equipped with a beam winding-engine, with a vertical cylinder 32 inches in diameter and 5 feet stroke, and two flat-rope drums. This engine formerly drew coals during the day, and at night it was coupled to a pumping-beam attached to a set of spears working a column of pumps, with a bucket 15 inches in diameter, raising water from the lodgment at the bottom of the shaft to a water-level 60 feet above, along which the water flowed to the Isabella shaft, where it was raised to the surface by the main pumping-engine at that pit. This engine is now used only for winding coals, the pumping being done by another engine.

The Isabella shaft is 429 feet deep, and is tubbed for a height of 165 feet, commencing at a depth of 120 feet from the surface, and ending at a height of 144 feet above the Beeston coal-seam. It was provided with a beam winding-engine, with a vertical cylinder 30 inches in diameter and 4½ feet stroke, spur-gearing, and two flat-rope drums. In November, 1891, this engine was replaced by an horizontal engine with two cylinders, each 20 inches in diameter and 3½ feet stroke, and a flat drum 12 feet in diameter. The writer has already remarked that there was a main pumping-engine at this shaft for dealing with the mine-feeders; this engine has a vertical cylinder 52 inches in diameter and 7½ feet stroke; there are three beams, one from the cylinder to the crank-shaft, and two from the spur-wheel shaft to the pump-spears in the shaft. There are twin sets of pumps, the lower having a bucket-lift 14½ inches in diameter, and the upper a forcing-set with a ram 14 inches in diameter.

The tubbing in the Sisters' shaft appears to have been originally ¾ inch thick, and of an equal thickness from top to bottom; the segments

being 2 feet deep by 3 feet 10½ inches long, strengthened by one bracket, and nine to the circle. (Figs. 1, 2, and 3, Plate IX.) This tubbing had not been very skilfully placed in position, many of the closing joints of wood being from 3 to 4 inches thick.

On December 27th, 1872, a portion of one of the joints of the tubbing in the Sisters' shaft was blown out, and the water rushed into the pit in enormous quantities, completely overpowering the pumping-engine, and the mine was speedily flooded. Owing to the heavy pressure of water, the greatest difficulty was found in approaching the leak. Many methods were tried, but it was not until the level of the water behind the tubbing had been greatly reduced that it was stopped. This was accomplished by forming a temporary platform across the shaft; upon this was firmly secured a hydraulic-jack which forced a narrow metal segment against the breach, and this was kept in position until the circle of the shaft was completed and wedged tight. By this time the water had risen to within a few feet of the breach. The workings of both pits were completely submerged, and their draining was a long and expensive process. The work was, however, executed, and the pits resumed work in the latter part of 1873.

In 1879, a system of longwall working was introduced; after about 4 acres of coal had been worked, the roof weighted very heavily, a break occurred which evidently reached the bottom of the sandstone, and a large feeder of water broke away, overpowering the pumping-engine at the Sisters' pit, and flooding the mine. The water gradually rose in the shaft, and finally passed along the water-level drift to the Isabella shaft, where it was pumped to the surface; this engine was fortunately just able to deal with its own and the additional influx of water, and thus prevented the flooding of the Isabella pit. After running for some months, the water-feeder gradually diminished so as to permit of the workings of the Sisters' pit being drained, and work was resumed.

Another longwall district of working was commenced later, and precisely the same event occurred, the workings of the Sisters' pit being again flooded.

In 1881, the owners were advised to erect an independent pumping-engine at the Sisters' pit, capable of dealing with the water-feeders at this pit, and of assisting the main pumping-engine at the Isabella pit, if required. The writer became manager of the collieries in August, 1881, and the work was carried out under his supervision. A Hathorn-Davey differential condensing pumping-engine was erected, capable of raising 630 gallons of water per minute from a depth of 540 feet

to the surface. The engine was placed in a suitable chamber prepared in the side of the shaft near the entrance of the old water-level. A staple was sunk to the Beeston lodge or water-standage so as to be quite independent of the working-shaft. The cylinder of this engine is 36 inches in diameter by 6 feet stroke, working two double-acting ram-pumps; and also by means of two quadrants, working two bucket-pumps, 12½ inches in diameter, placed in the staple, by which the water is raised to a tank in the engine-house, whence it is forced to the surface by the ram-pumps.

After the Sisters' pit workings were cleared from water, they were put into order, extended, and the output from the two pits increased from 85,000 tons in 1873 to 104,000 tons per annum. After the inundation of the workings in 1872 the owners saw how necessary it was to have a shaft near the outcrop, at which coal could be wrought in cases of emergency. The Lily pit was sunk to the Beeston seam in June, 1875; it is fitted up with a winding-engine, boilers, heapstead, etc., and has been found very useful on the occasions when the other pits were flooded in 1881 and 1883.

On March 8th, 1883, at 9 a.m., without any warning or indication of weakness, the tubbing in the Sisters' shaft suddenly collapsed at a depth of 336 feet from the surface, 216 feet from the top of the tubbing, and a feeder of water, estimated at 4,000 gallons per minute, rushed into the pit. The men and horses were taken out by the Isabella shaft, and the whole of the Sisters' pit workings were quickly drowned out; the water also rapidly made its way into the workings of the Isabella pit, and in a few days the whole of the workings were under water, which commenced rising up the shafts. On examining the breach it was found that an irregular hole, of about 160 square inches in area had been broken out of one of the segments (Fig. 1, Plate IX.). Strenuous efforts were made to stop the breach by means of iron plates, plugs, etc., but the metal was deteriorated and the edges kept breaking away. In the meantime, the apparatus shown in Figs. 7 and 8 (Plate IX.) was prepared: it consisted of a duplex lever, the extended arms being fitted with right and left-hand screws, iron plates of the same radius as the shaft-tubbing being fixed at the ends *A* and *B*; these plates were 4 feet long, 3 feet deep, and 1¼ inches thick, and were covered with a thick indiarubber pad. The apparatus was brought opposite to the breach, the screw was turned, and forced the iron plates against the hole. The pressure and volume of the issuing water, however, was so great that at first the indiarubber was ripped up like paper; ultimately it was protected by copper wire-netting. On

the second inlet the apparatus was screwed up and the pad forced securely against the breach. At first the pad appeared to be perfectly successful, but unfortunately the pressure of the screw had forced back and cracked the adjoining segments of tubbing, and the appliances had to be abandoned. By this time the water had risen to the level of the Hatfield-Davey pumping-engine, which was very quickly submerged, and the water continued to rise gradually up the shafts, nothing further could then be done. The pad plan of stopping the breach would have been perfectly successful, but for the giving way of the adjoining segments of the tubbing, the cause of which will be afterwards explained.

A scheme was then prepared for the unwatering of the collieries and the re-tubbing of the Sisters' shaft, which met with the sanction and approval of the owners, and was ultimately carried to a successful issue.

It was decided to keep the main pumping-engine at the Isabella pit running to the full extent of its capacity: to fit up and attach to the Sisters' pit winding-engine a lifting set of pumps 20 inches in diameter to be slung in the shaft: and to equip the Isabella and Lily pits with tanks for drawing water to the utmost power of each winding-engine. These tanks were fitted with automatic filling and delivery-arrangements so as to require as little labour as possible. With these appliances 2,200 gallons of water per minute could be raised, a quantity somewhat in excess of the sandstone feeder.

While this work was in progress preparations were made to take out and renew the tubbing. The upper portion yet uncovered by water was found to be strong and sound, and it was decided to commence with the new tubbing 99 feet below its top: no crib bed being available, the following method was adopted to carry the 99 feet of old tubbing. No. 1 segment of the ring of tubbing was taken out, two holes 1½ inches in diameter were drilled 3 feet deep into the stone, and two iron pins or dowels were driven firmly into the holes, closely adjacent to the bottom of the upper segment of the tubbing. No. 2 segment was then removed, and two similar pins were inserted, and so on until the entire ring of tubbing was removed; and the weight of the 99 feet of old tubbing was found to be safely and effectually carried by the pins or dowels.

By this time, the combined efforts of the pumping and winding-engines were taking effect, the water being slowly reduced. As the water lowered, the old tubbing was removed, a bed secured, a wedging-crib inserted, and the new tubbing carried up to the first tier, which was supported by the iron pins, making the shaft perfectly safe to this point.

The tubbing put into the Sisters' pit in 1883 consists of segments 4 feet 1½ inches long, 2 feet deep, 4 inches on the bed, and 1¼ inches thick. Between both horizontal and vertical joints, yellow-pine sheeting ½ inch thick was inserted; this was afterwards filled with red-pine wedges. The wedging-cribs were 6 inches deep and 20 inches on the face; two segments in each wedging-crib were fitted with valves, and stand-pipes were carried from one length to the other, to carry away any air that might have been accumulated. Behind, each wedging-crib was glutted with wood, filled full of wedges, and thin concrete was run in so as to close any small cavities still remaining. As the segments were put on, they were properly stayed behind with wood, to the circle of the pit, and concrete was run in composed of limestone, marl, and cement.

The position of the old breach of 1872 was reached on August, 1883. The tubbing was found to be sound and in fairly good condition, but the ground behind the tubbing was found to be hollow, and so far as could be ascertained, the tubbing had not been properly packed and stayed. The shaft was found to be in this condition down to and below the point where the tubbing burst on March 8th, 1883. The point of fracture was just above the Barcelona coal-seam and the cause of the failure of the tubbing and the giving way of the segments under the pressure of the screw and plates was clearly demonstrated. In the seam and for a short distance above and below it, the back of the old tubbing was found to be extensively corroded, the metal was soft, in some places the thickness was about ⅛ inch, the point of a pick could be easily driven through it, and the tubbing appeared as if it had been submitted to the action of an acid water.

At the Barcelona coal-seam, an immense quantity of soft *débris* had been washed out, a space being formed in some places at least 20 feet wide. Fortunately there was a wedging-crib just below, and after spiling behind it with timbers to hinder it from shifting, a lift of tubbing was speedily inserted, and the cavity filled with cement concrete. The size of the opening may be gathered from the fact that 150 tons of concrete were used. It may be remarked that had the tubbing been properly backed with concrete or other material past this seam, the fracture could not have taken place.

Below this point the water-feeder was collected in pipes, and run into the shaft, as circle by circle of the tubbing was taken out, until the main crib-bed was reached. This crib-bed was perfect and tight, and the tubbing was closed and made secure on September 20th, 1883.

One or two matters of interest may be mentioned, viz., the slinging of the 20-inches set of pumps in the Sisters' shaft was done without the use of ground-crabs, the pumps being suspended by rods screwed at the upper

ends. When a depth of 270 feet had been reached it was found that the limit of the power of the engine, with 20 inches pumps, had been reached. A diminishing piece was thereupon fixed, 15 inches pumps were attached, and pumping was continued until the water was lowered below the level of the Hathorn-Davey pumping-engine chamber. The length of the sett was 360 feet, probably the greatest depth at which a lifting-sett had ever been worked.

When the Hathorn-Davey pumping-engine was uncovered, and the water run off, it was found uninjured, although slightly coated with mud, and commenced working as soon as steam was turned on.

The time occupied by all these operations extended over twenty-three weeks, and the pits to the rise began drawing coals on September 12th, 1883.

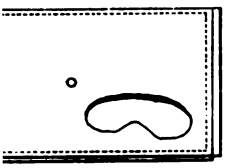
The roads and workings were much damaged by the water, and it was some time before the output reached the quantity raised previous to the stoppage, viz., 104,000 tons per annum. In 1884, the output was 92,000 tons; in 1885, 138,000 tons; and in 1894, 250,000 tons, or $2\frac{3}{4}$ times the output at the time the writer took charge of the collieries.

In the Isabella shaft, tubbing was inserted in a similar fashion to that in the Sisters' shaft—there are four thick rings of wooden sheeting. On February 7th, 1887, during a very keen frost a piece of the sheeting blew out, 16 inches long by 3 inches thick, in exactly the same manner as in the case of the Sisters' shaft in 1872. Owing to the great pressure of water it was impossible to face it. Wooden battens, 8 inches by 3 inches, were with great difficulty forced into the hole, and the inflow was stopped in two hours. After this occurrence, wooden cribs were placed inside the shaft in front of the thick sheeting, and since then there has been no further trouble with the joints.

Mr. A. LUPTON (Leeds) said that having managed an adjoining colliery where 4,000 gallons of water per minute had been raised for some time, he knew what difficulties nature could put in the way of the production of coal. According to the rules laid down in Mr. Greenwell's treatise on *Mine Engineering*, the tubbing at Garforth collieries ought to have been 1 inch, or a little more, in thickness, whereas it had been only $\frac{3}{4}$ inch thick, so that their predecessors had not erred on the side of safety.

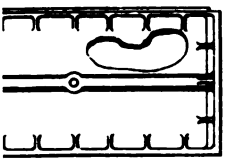
Mr. JOHN GERRARD (H.M. Inspector of Mines, Manchester), said that Mr. Routledge's paper was a thoroughly practical account of the

To illustrate Mr R. Routledge's Paper on "The Garforth Collieries" etc.

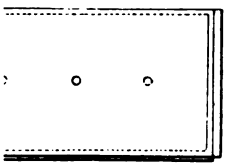


G. 1.—FRONT VIEW.

FIG. 2.—PLAN.

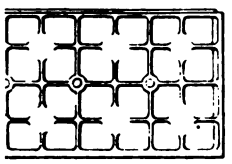


3. 3.—BACK VIEW.



1. 4.—FRONT VIEW.

FIG. 5.—PLAN.



6 -BACK VIEW

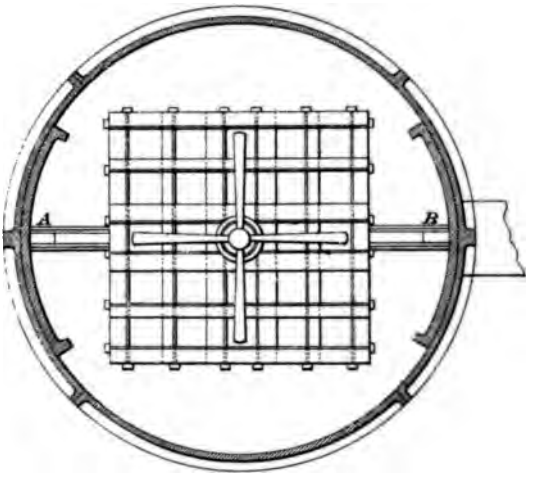


FIG. 7.

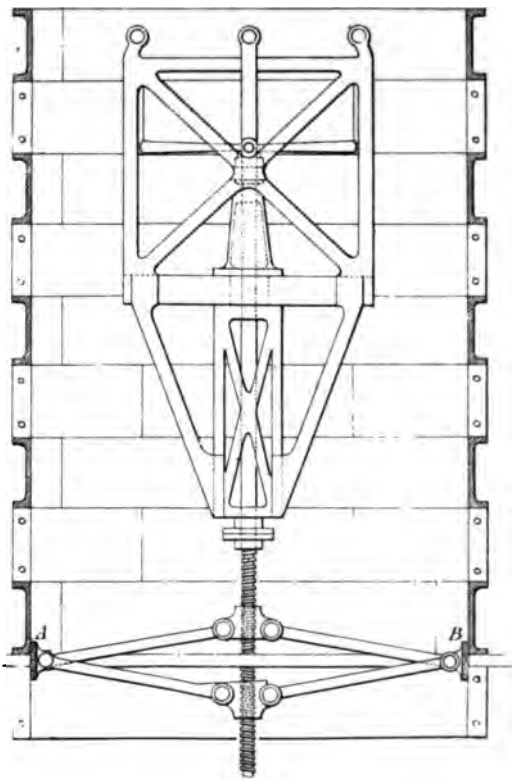


FIG. 8.

means adopted to overcome difficulties which had also arisen at other collieries, and it would be very valuable for reference. There was a considerable depth of tubbing in the two shafts at Shireoaks colliery, and more than one failure of segments of tubbing had occurred. It would be interesting if some one would describe the occurrence and mode of dealing with the fractures of the tubbing at the Shireoaks colliery.

Mr. ROUTLEDGE said that at Ryhope colliery, where they had the greatest depth of tubbing of any colliery in the North of England, 4 inches of wooden sheeting had been inserted; it had been blown out on several occasions, and metal was inserted instead. The circumstances were, however, somewhat different, as at Ryhope the wedging-cribs were placed at intervals of 60 feet. When the accident occurred at Garforth colliery, none of the wedging-cribs were tight, and the entire pressure of water had to be overcome. At Ryhope colliery, where wedging-cribs were put in, the pressure was not greater than that between two wedging-cribs.

Mr. W. E. GARFORTH (Normanton) said it would be interesting if, in addition to the sketches of tubbing Mr. Routledge would append to his paper the actual dimensions and thicknesses of the tubbing, and describe the kind and size of the wedges, and other matters of detail, which made all the difference between success and failure in an important matter of this kind. Could he also append a short account of how he dealt with the cavities behind the tubbing, and the sort of filling used; and whether means had been taken for preventing corrosion of the tubbing by Dr. Angus Smith's composition or any other method, and what means he took to discover the weak tubbing (as the strength of the tubbing was only equal to the weakest segment). Had the water been analysed?*

The PRESIDENT said that wooden cribs were placed inside the shaft, and asked whether metal would not have been better?

Mr. ROUTLEDGE (Garforth colliery) said he thought not. The wooden cribs were 18 inches thick, set with cement behind, and they became like a stone wall. The wooden cribs referred to were put in the Isabella shaft, in front of some thick sheeting, to prevent a similar occurrence as that which occurred in the Sisters' shaft in 1872. The tubbing was now $1\frac{1}{4}$ inches in thickness, and was well protected at the back. When they came up with a length, from one wedging-crib to another, instead of inserting a thick wooden sheet a metal closer was put in, the joint was filled with $\frac{1}{2}$ inch sheeting between the vertical and horizontal joints, filled in with wooden wedging. All the tubbing was backed with

* Mr. Routledge has made these suggested additions to his paper.

concrete. He had a piece of tubbing $\frac{1}{8}$ inch thick, and a lump of sandstone weighing about 28 pounds that had come through a hole in the tubbing.

Mr. NASH asked what was the thickness of strata between the sandstone and the coal?

Mr. ROUTLEDGE replied that it was 180 feet. He added that every segment of the tubbing was tried and tested, and 140 pieces were rejected.

Mr. LUPTON asked whether, since this breakdown, any pillars had been taken out or any longwall worked, and whether any other breakdowns had occurred?

Mr. ROUTLEDGE said that all the workings were upon the bord-and-pillar system since the occurrence described in his paper. A large quantity of coal had been worked out without any break whatever. The largest break was accompanied with a feeder of 320 gallons per minute in the dip workings. The compressed-air pumping plant was carried to the face of the workings, otherwise the colliery would have been drowned out on two or three occasions.

Mr. A. M. CHAMBERS (Thorncliffe Collieries) moved a vote of thanks to Mr. Routledge for his paper.

Mr. J. GERRARD seconded the motion.

The vote of thanks was agreed to, and briefly acknowledged by Mr. Routledge.

The following paper by the PRESIDENT (Mr. John Nevin), on "The Mexican or Patio Process of reducing Silver Ores," was then read :—

THE MEXICAN OR PATIO PROCESS OF REDUCING SILVER ORES.

By JOHN NEVIN.

Introduction.

The modern methods of treating silver as well as gold ores have been described in several very able papers read before the North of England Institute of Mining and Mechanical Engineers by Mr. A. G. Charleton* and by Mr. C. A. Stetefeldt,† but the writer having just returned from a third visit to Mexico, has thought that some description of the method by which the greater part of the silver ores worked in that country have been for three centuries, and to a great extent are yet reduced, might be of interest to the members.

Unfortunately, at present, silver seems to be rapidly approaching the point of little value that it at one time reached in Judea, 1 Kings xi., 21 (it was nothing accounted of in the days of Solomon), but in Mexico and some parts of the United States of America its production is still one of the chief industries.

Mexico is the second largest silver-producing country in the world, the United States of America coming first. The greater part of the silver and gold obtained in Mexico is coined at the various mints there. It is estimated that since the establishment of these mints in 1537, for 350 years up to July, 1887, the total coinage has been as follows:—

	Years.	Gold. Dollars.	Silver. Dollars.	Copper. Dollars.
Colonial epoch—1537-1821	284	68,778,411	2,082,260,656	542,893
Independence—1822-1873	52	45,598,020	758,822,054	5,235,177
Republic—1873-1887	14	8,045,749	323,883,608	203,296
Totals	350	122,422,180	3,164,966,318	5,981,366

The production since then has shown a steady increase, and is estimated to have been—

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351; vol. v., page 271; vol. vi., pages 69, 295, and 457; and vol. vii., page 67.

† *Trans. Fed. Inst.*, vol. viii., page 93.

	Gold. Dollars.	Silver. Dollars.
1887-8	1,031,000 ...	34,912,000
1888-9	1,040,000 ...	40,706,000
1889-90	1,100,000 ...	41,500,000
1890-1	1,150,000 ...	43,000,000
1891-2	1,275,000 ...	45,750,000
Totals	5,596,000	205,868,000

The world's production for the same period is estimated as follows:—

	Gold. Dollars.	Silver. Dollars.
1887	105,774,955 ...	124,280,978
1888	110,196,915 ...	140,706,413
1889	120,971,514 ...	158,759,468
1890	120,475,300 ...	173,743,000
1891	125,299,700 ...	185,599,600
Totals ...	582,718,384	783,089,459

For these five years, therefore, Mexico produced 0·96 per cent. of the gold and 26·29 per cent. of the silver output of the whole world. These values for the last five years are estimated in United States currency, and are taken from *The Mineral Industry . . . . in the United States . . .* 1892.

Most of the silver-mines in Mexico are situated on the great plateau or table-land which occupies the greater part of the interior of that country, reaching from latitude 19 degs. north to the boundary-line of the United States of America at latitude 32 degs. north; or in the hills which border it. This plateau varies from 5,000 to 6,000 feet above sea-level, and the hills rise to from 7,000 to 10,000 feet, especially on the west, where they form the Sierra Madre, a continuation of the chain of the Rocky Mountains. Its climate is sub-tropical, the greater part of it being called *tierra templada* or temperate land, to distinguish it from the *tierra caliente*, the hot land near the coast. The rainfall is small, from 12 to 16 inches per year, and this nearly all falls in the three months from June to September, a shower in any other of the nine months being of rare occurrence. The heat in the middle of the day will sometimes reach 80 or 90 degs. Fahr. in the shade, but the nights are always cool, and in winter sometimes frosty.

Owing to the small rainfall and its unequal distribution the bulk of the plateau for the greater part of the year is practically desert-land. Part of it, however, is ploughed and a crop taken off after the rainy season. In the river valleys where water is obtainable for irrigation, or where reservoirs have been made in the hills to store it for that purpose,

crops can be grown nearly all the year round; but this is a very small fraction of the whole, and, except in the mountains where oak and pine timber grow, the rest of the country is bare of wood and covered over with small mesquite scrub and cactus of various kinds.

Although coal has been found in Sonora* and other parts of the country it has up to the present only been worked on a commercial scale in the state of Coahuila and that only within the last few years. It is only within the last ten or twelve years that the Mexican Central and International Railways have enabled coal to be be imported to the mining districts. The scarcity of water and fuel, the great drawback of Mexico, has therefore to be taken into account in choosing any process for treating the ores.

The geology is very varied and intricate: igneous action has been both extensive and frequent. In the states of Zacatecas (about Fresnillo and Nieves), and in part of Durango, with which the writer is personally acquainted, the prevailing rocks are chiefly clay-slates, with gritty and calcareous beds among them, probably of Silurian or Devonian age; the higher hills are often capped with a porphyritic granite which alters the slates at the point of contact, and dykes and veins of basalt and other volcanic rocks are frequent.

Further north, near Symon and Torreon, are hills of a compact blue limestone which may possibly be of Carboniferous age, but the writer has so far been unable to find any fossils.

Some of the principal mining centres are Guanajuato, in the state of that name; Catorce, in San Luis Potosi; Real del Monte, near Mexico City; Zacatecas, Sombrerete, and Fresnillo in the state of Zacatecas; Parral, in Durango; and the Sierra Mojada, in Coahuila, where much galena is found.

Although the country is generally almost waterless at the surface, water is generally met with in the mines at a depth of 100 to 300 feet sometimes in large quantities, and the difficulty of dealing with it by the power of men and animals only (*fuerza de sangre*) has prevented most of the mines from being worked much below water-level; when additional railways make coal cheaper and more generally available there will be a great expansion in Mexican mining.

A good account of native Mexican mining and miners, and the various difficulties to be encountered, is given in Mr. W. H. Glennie's "Notes on

* "The Coal-fields of Sonora, Mexico," by Mr. J. Overend, *Trans. Fed. Inst.*, vol. vii., page 230.

Mining in North Mexico."* There are, however, many mines worked by English, American, and Mexican companies on modern systems and with good machinery.

The ores of silver found comprise nearly every variety.

About Zacatecas native silver, stephanite, brittle or antimonial silver, argentite, silver glance or sulphide, and pyrargyrite or ruby silver, are found; some of these ores also contain gold.

Near Nieves the ores are oxidized near the surface, and are chiefly sulphide or antimonial silver at lower levels. Native silver has not been found.

Near Symon are rich mines of cerargyrite or chloride of silver, and embolite or chloro-bromide of silver in limestone, obtained both from veins and *mantos* or horizontal layers. Iodide of silver has also been found.

Argentiferous galena is also common, and is of great importance as helping to smelt the drier ores.

When ores are sufficiently rich in lead, or lead-ore can be obtained to mix with them, and fuel is available, they are smelted, but by far the greater portion is treated by amalgamation.

The patio process of reduction is said to have been invented in 1557 by Bartolomé de Medina, a priest of Pachuca, near Mexico City, and is first mentioned in a report to the Viceroy of Mexico by Barrio de Montalvo in 1643.

The Patio Process.

The ore as it comes from the mine is generally sorted near the top of the shaft, and broken into pieces not exceeding the size of a hen's egg. In this state it is carried to the *hacienda de beneficio* or reduction-works: these consist of a series of buildings containing the various machinery, stores, etc., placed round one or more large stone-paved courtyards or patios (Fig. 1, Plate X.). The ore is then crushed in a *molino* or mill; usually this is in the form of a Chilian wheel, consisting of a circular stone-paved dish about 6 to 12 inches deep, and 10 to 12 feet in diameter, with a wooden or stone post in the centre 3 to 5 feet high. To this is pivoted a horizontal shaft or axle on which runs a vertical stone roller, 6 to 10 feet in diameter and 12 to 18 inches wide; the outer end of the shaft projects beyond the circular path of the roller and one or more mules or horses are harnessed to it, and by going round in a circle cause the roller to crush the ore. One of these mills with nine mules, working three at a time for three hours, will crush about 6 tons in twenty-four hours,

* *Trans. Fed. Inst.*, vol. i., page 39.

or one with a cast-iron bottom will crush 8 tons, but the presence in the latter case of metallic iron in the crushed ore is said sometimes to interfere with the reduction. Occasionally a stamp-mill is used, generally of eight stamps, with wrought-iron heads and wooden sterns, driven by spur-wheels, actuated by mules or horses. A mill of this kind driven by twelve mules, three at a time, will crush 4 to 6 tons in twenty-four hours, and for labour will require in that time two *arreadores* or drivers, and five or six boys to feed and carry away the ore.

The *granza*, or crushed ore from the stamps is then taken to the *arrastras* or *tahonas*, where water is added. These consist of a circular stone pavement about 12 feet in diameter, round which is a curb of wood or stone forming a water-tight tub about 2 feet deep: in the middle is a block of wood or stone containing a footstep in which works the iron pivot on the foot of an upright shaft, the top of which is carried by a journal on a horizontal beam. This upright shaft carries two or four horizontal arms to the outside ends of which mules or horses are harnessed, and from each of these is hung, by two cords or chains, a *voladero* or runner of stone, 3 to 4 feet long, 1½ to 2 feet wide, and 12 to 15 inches thick, which drags over the stone bottom and so grinds the ore (in some cases cast-iron bottoms are used).

An ordinary *arrastra* driven by four mules, working two at a time for six hour shifts, will grind from ½ ton to 1½ tons in twenty-four hours; at Fresnillo, when driven by steam-power, a 12 feet *arrastra* will grind 1½ tons, and one 16 feet in diameter 3 tons in eight hours. Two men, or one man and one boy will look after and charge five ordinary *arrastras* in the daytime, and one man will look after ten or twelve at night. The water used is generally one and a half times the weight of the ore.

When sufficiently ground, the liquid ore now called *lama* is either ladled out of the *arrastras* into barrels and carried, or run out through spouts, into the *lameros* or settling-tanks, where it partially dries. When a sufficient quantity has been collected to form a *torta*, it is taken out on to the patio and placed in a semi-liquid state between planks made tight with horse-dung, where under the clear sky and rarefied air it soon dries sufficiently to be ready for treating. A *torta* may contain any quantity from 15 to 80 tons, according to quality of ore, time of year, size of plant, or other circumstances. A common size is about 50 tons, which when ready for treading will occupy a circle of about 50 feet in diameter and 10 to 12 inches thick.

The operations are now taken in charge of by the *azoguero* or amalgamator (a word derived from *azogue* or mercury).

The first proceeding is to sprinkle salt over the *torta*, in quan varying from 2½ to 6 per cent. of the weight of ore. This salt is so times obtained from the coast, but generally from inland salt-lakes which case it is often very impure. The *torta* is then turned over v shovels, and has its first *repaso*, or treading by eight to fifteen anim for 4 or 5 hours; one or two men with bare feet and legs, standing i the middle, hold cords attached to the halters of the mules or horses, drive them in circles commencing from the outside of the *torta* working towards the centre, and taking care that every part of it shal well trodden and mixed.

Twenty-four hours after this, *magistral* is added. This is prep by slowly roasting in a reverberatory furnace a mixture of copper iron-pyrites to which a little salt has been added. The sulphate of per formed by this process generally amounts to between 20 and 40 cent. of the whole, and is the chief agent in reducing the ore, altho the salts of iron assist. The quantity of *magistral* used varies, accordin its quality and the variety of ore, but is generally from 1 to 2 per cen the weight of the ore. Sometimes sulphate of copper is used instea *magistral*, when 5 lbs. to the ton of ore is generally sufficient. *magistral* is used in winter than in summer. After adding the *magis* the *torta* is trodden for an hour, and then the first quantity of merc called the *encorporo*, is added; this is generally about two-thirds of quantity intended to be used for the reduction, which is usually abou times the weight of the silver expected. The mercury is squ through a canvas bag so that it may fall on the *torta* in finely-divi globules, and a further *repaso* takes place immediately afterwa Sometimes the salt, *magistral*, and first quantity of mercury are adde the *torta* together.

After the salt, *magistral* and mercury have been added to mixed with the *torta*, chemical action commences, and is wat carefully by the amalgamator by means of trials of samples (*tentadu* of about 8 ounces each, taken from twenty or thirty different p of the *torta*, and washed in a *jicara* or vanning-horn. This has compared to a mirror in which the *azoguero* can see what is going o the *torta*. On the first trial, soon after the *repaso*, on stirring the tents of the spoon with the fingers, and agitating it at the surface vessel of water, the light dirt is washed off, and the remaining met substances and mercury are closely examined. At this stage the cury contains but little silver, and its colour and state of division the only indications of the working of the *torta*. If the colour is nat

or slightly bronzed, the reaction is too slow, the *torta* is cold, and more *magistral* is wanted; if the surface is moderately grey, all is going on well, but if it is dark grey and the upper part of the *tentadura* shows ashy-grey powder, *desecho*, which will not unite into globules, the reaction is too rapid, the *torta* is hot, the *repasos* must be stopped, and lime or ashes added to reduce the loss of mercury. Generally, the colour of the mercury guides the proportion of *magistral*, the condition of the *limadura* indicates the daily progress, and the solidity of the amalgam determines the addition of mercury and the end of the operation.

Twenty-four hours after the *encorporo*, if the proportion of *magistral* is right, and the process doing well, amalgam will be found in the mercury, little *desecho* is seen, and the *limadura* is coarser and more consistent, and it can be compressed by the thumb into *pastillas* or lozenges.

When the *limadura* becomes dry, and will not give a drop when pressed by the thumb, more mercury is added to the *torta* in small quantities, this is called *cebar* (to feed it).

The whole process lasts from fifteen to forty days, according to the quality of the ore and the season of the year, taking longer in winter. The *repasos* required vary in number from eight to twenty or twenty-five, and generally take place every two days, or every day if required, a *torta* of 50 tons requiring six or eight animals for six hours at a time. They are more frequent in winter than in summer, and they hasten the process when frequently used. In an experiment, two *tortas* of the same ore were put on the patio at the same time, one with eight *repasos* of five or six hours at intervals of three or four days took twenty-seven days, the other trodden day and night by the same number of animals took eighty hours. The yield of silver and loss of mercury were about the same, but the increased speed did not pay for the extra cost of labour.

When the chemical action has ceased, and the silver which can be extracted has been taken up by the mercury, the *torta* is then said to be *rendido* and the operation is finished.

The chemical reactions which take place during the process are complicated and difficult to understand. The ore contains native silver, chloride of silver, and the sulphides of various metals containing silver; the general result is that the sulphate of copper and chloride of sodium (salt) react on each other and on the sulphide of silver, converting the latter into chloride, in which form it can be reduced by the metallic mercury.

In the Zacatecas district, when the *torta* is finished (*rendido*), a further quantity of mercury, called the *baño* or bath, often amounting to

four-fifths of the quantity already used, is generally added, and after or more *repaso* the ore is carried in tubs or hand-barrows to the *lavaderos* (washers (circular tanks of wood or masonry with the bottom formed (one stone). These tanks vary in size, but are often about 9 feet in diamet(by 6 feet deep; they are fitted with vertical shafts carrying stirrers, th stirrers of two tanks being driven by a mule-gin. Each tank will was 2½ tons per hour, the ore is put in, in small quantities, water being run i at the same time and the stirrers kept revolving. The mud and wat(are tapped off through holes in the sides, and the amalgam, which, owin to the quantity of mercury added in the *baño*, is very liquid is left in th bottom and ladled out into bowls or *bateas*. A good deal of amalgam go(away in the slimes from the washers, and these are re-washed by hand o *planillas* or sloping-tables, but much is still lost. At Guanajuato, th ore passes through three washers, one after the other, which is a bett(system and saves some of this waste.

The amalgam from the washer is taken to a room called the *azogueri* where it is filtered through a strong leather bag (*manga*) with a canva bottom, attached at the top to an iron ring suspended from a beam. Th mercury gradually passes through the canvas into a bowl beneath, leavin in the *manga* a pasty mass of amalgam still containing four to five par of mercury to one of silver. This is emptied out of the *manga* on to leather-covered table and pressed into moulds about 3 inches thick, (such a shape that six of them when put together form a circular cak(with a hole through the middle. Each piece (*marquesta*) will weigh abou 20 pounds. These cakes are piled up into a column on a grating und(a copper or cast-iron bell (*capellina*) about 3 feet high, 18 inches i diameter, and 1½ inches thick, which is raised or lowered by a capstan the base of this fits into a hollow in a stone or iron base-plate throug which cold water is constantly kept running. An annular fire-place (stone, adobe, or iron, is built round the bell and filled with wood (charcoal, which is fired for from eight to twenty-four hours. As the he(volatilizes the mercury, it is condensed by the cold water and ru through an iron pipe into a special reservoir.

At a small hacienda, which the writer visited in February last, th amalgam was retorted in the iron flasks in which mercury is purchase(holding about 76 lbs. each. The bottom of the flask was cut off an a pipe screwed into the mouth, they were filled with amalgam and a li carefully luted on; they were then placed in a small iron fire-place, wit the pipe inserted in an earthenware vessel (*olla*) kept full of cold wate in which the mercury condensed and was collected; the loss was said t be very small (Fig. 2.)

From 2,000 lbs. of amalgam one should obtain 400 lbs. of si almost free from mercury. The silver forms a porous spongy mass; generally very pure, running over 990 fine; it is fused and cast into bars, and the greater part of it is taken to the various mints, where it is coined into Mexican dollars.

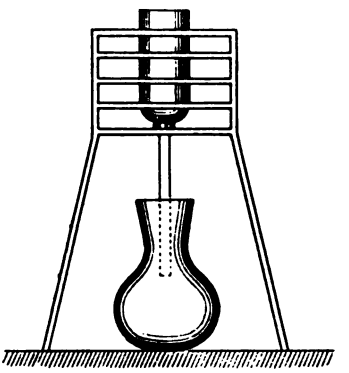


Fig. 2.

The loss of mercury in retorting, when carefully done, is very small, sometimes as low as 1 oz. in 100 lbs. Out of a batch of 170 lbs. of amalgam which the writer saw retorted in flasks in February, 1894, the loss was about 4 ounces.

One of the largest establishments which was ever put down for the patio process is the Hacienda Grande at Fresnillo. These works were built about 1830, at a cost of over 300,000 dollars. The hacienda is supplied by water pumped from the Proaño mines, about half a mile distant. On each side of the main gateway are the offices and the residences of the managers. The main *patio* will hold 64 *tortas* of 60 tons each; a fountain in the middle supplies water, which is carried by conduits to the *lavaderos* and *tentaduras*. Round this patio, on three sides, are galleries which can and have contained 314 *arrastras*; in the middle of each line of *arrastras* are the *lavaderos*, the stirrers of the two vats being turned by horse-whims. In close vicinity are the *azoguerias*, where the amalgam is filtered and the *quemaderias* with open roofs, under which are the *capellinas*, or bells for distilling the mercury. Behind the *arrastras* were at one time 12 stamp-mills. An outer court contained store-rooms and stables for 1,500 animals. A third court contained the *magistral*-furnaces and *planillas*, or slime-washers (Fig. 1, Plate X.)

The crushing machinery is now driven by two Cornish engines, with cylinders 40 inches in diameter and 9 feet stroke, made in England, sent by sea to Vera Cruz, and brought up the country by road. These engines are said to have cost their weight in silver. Each engine has three Cornish boilers fired with Coahuila coal.

In 1838 at these works, 28,047 *montones*, or tons, of ore were worked, yielding 229,035 marks, or 1,832,280 ounces of silver (65·28 ounces per ton), at a cost of 645,370 dollars, or 23 dollars per ton, made up as follows:—

	Dollars.
Forage and shoeing mules and horses	76,940
Animals died	484
Mercury, 199,043 lbs. at 1·10 dollars	218,947
Magistral, 11,439 cargas	59,218
Salt, 77,718 fanegas of 200 lbs.	111,287
Lime, 465 fanegas, at 31¼ cents	145
Fuel—Wood, 90,685 arrobas of 25 lbs, at 6¼ cents; charcoal, 6,042 arrobas, at 18¾ cents	6,801
Stone for arrastras, etc.	3,908
Iron and bronze	10,956
Candles and stores	6,018
Labourers	79,733
Fixed salaries	32,244
Half management, other half charged to mine	16,112
Board and lodging	7,577
Interest on 300,000 dollars at 5 per cent.	15,000
Total	645,370

In the years 1840, 1841, and nine months of 1842, these works treated 95,570 tons of ore, yielding 537,250 marks of silver valued at 4,700,920 dollars, at a cost of 1,900,081 dollars, the average value of silver recovered from the ore being 45 ozs., or 49 dollars per ton, and the cost 19·88 dollars per ton. The mercury used varied from 6·35 to 5·75 dollars per ton of ore, and from 14 to 11½ ozs. per mark of silver recovered, the other costs varied from 14·43 to 11·77 dollars per ton, the total cost being from 20·78 to 17·52 dollars, of which the mercury formed about one-third. The loss of silver during this period is said to have been about 25, but in some years as low as 10 per cent. Although 25 per cent. seems a very high loss, the Washoe process in the United States on some ores loses 30 per cent. Messrs. Ortega, who are at present working this hacienda, have introduced improvements by which the process now only takes twenty-four hours on the patio, instead of as many days, and the cost of treatment is said to be under 6 dollars per ton; they are now treating 100 to 150 tons a day of ores chiefly from the dumps of the Proaño mines, which have been standing for some years, but are likely to be soon restarted.

This process of amalgamation, carried on as it is in the open-air, does not often affect the health of the workpeople; in spite of the quantity of mercury used the *repasadores* and *lavadores* suffer no inconvenience. The men who filter the amalgam sometimes experience nervous irritation, but no serious consequences ensue. The *quemadores* or retorters, if a belt bursts, which rarely happens, suffer from severe salivation and nervous trembling, recovering therefrom with difficulty.

Horses and mules are fond of the pulp and often lick it, probably for the salt it contains, and balls of amalgam are not seldom found in their stomachs after death. The copper salts discolour their feet and legs, and it is always easy to distinguish a horse or mule which has worked on the patio; they sometimes die from cracked legs and feet caused by these salts.

As transport by rail becomes more easily available, and fuel and water carried by it become more plentiful, the patio process will probably give way to more modern methods of reduction by mercury or leaching, but for 300 years it has proved itself well adapted to the conditions of the country. After the ore is pulverized, the silver is extracted with no apparatus but a washer and a bronze bell, no labour but treading by the feet of men or animals, no fuel except for roasting *magistral* and retorting amalgam, no re-agents except 2 to 6 per cent. of salt, and 1 to 3 per cent. of *magistral*, and with a loss of mercury varying from the weight of silver recovered to about twice that quantity.

APPENDIX.

Glossary of a few Spanish words used in Mexican mining and reduction-works:—

Adobe, sun-dried brick.
Arrastra, a machine for grinding ore.
Arreador, a driver of horses or mules.
Arroba, a weight of 25 lbs.
Azogue, mercury.
Azogueria, the place where the mercury is recovered from the amalgam.
Azoguero, the amalgamator, the man in charge of the reduction-process.
Azolaques, blue or sulphide ores.
Azul, blue.
Baño, a bath, the last quantity of mercury added to the *torta*.
Batea, a bowl or dish.
Beneficio, the process of reduction.
Bonanza, a mine is in *bonanza* when making profits.
Borrasca, a mine is in *borrasca* when not paying costs.
Caliente, warm, applied to a *torta* when not working properly.
Cantera, a quarry, quarry-stone, usually a kind of porphyry.
Capellina, a bell of metal under which amalgam is retorted.
Carga, a load of 300 lbs. weight.
Cata, a shallow surface-pit.
Cebar, to feed, to add mercury to the *torta*.
Colorado, red oxidized ore from shallow workings.
Cuchara, a vanning-spoon, a horn dish used for testing ores, etc.
Desecho, decomposed mercury found in the *tentadura*.
Encorporo, the mixing of the mercury and other re-agents with the ore.
Fanega, a measure containing 1·6 English bushels.
Granza, sand, ore after being crushed.
Grueso, large ore in distinction to small ore (*tierra*).
Guija, quartz or spar.
Hacienda, the homestead of an estate.
Hacienda de beneficio, reduction-works.
Hechado, the dip of a vein.
Jales, the pulp or refuse from the torta after washing.
Jicara, a cup, the horn vanning-spoon used for testing the *torta*.
Lama, mud, the ore after being ground with water in the *arrastra*.
Lamaja, clay.
Lamero, a settling-tank.
Lavadero, a washer or washing-tank.
Limadura, amalgam of mercury and silver.

Magistral, a preparation of copper and iron pyrites used in the *patio* process.
Manga, a bag of leather and canvas for straining amalgam.
Manto, a horizontal layer or bed of ore.
Marc, a weight of 8 ounces.
Marqueta, the mould of amalgam ready for retorting.
Metal, ore.
Molino, a mill, applied to either a Chilian wheel or stamp-mill.
Monton, a pile, a weight of a ton: at Zacatecas it is 20 quintals of 100 Spanish or 101½ English lbs.
Negro, black.
Onza, the Spanish ounce of 442·72 grains troy.
Palma, quarter of a vara, 8¼ inches.
Pastilla, the small adherent bits of amalgam in the *tentadura*.
Patio, a courtyard.
Planilla, the inclined plane upon which fine ores or tailings are washed.
Plata, silver.
Polvillos, finely divided ore or amalgam in the *tentadura*.
Pozo, a pit or winze.
Pulgada, a length of 1 inch.
Quemador, a burner, the man who retorts the amalgam.
Quintal, a weight of 100 Spanish or 101½ English lbs.
Rendido, subdued, said of a *torta* when reduction is finished.
Repasador, the man in charge of the treading.
Repaso, the treading of the *torta* by animals or men.
Tahona, an *arrastra*.
Templada, temperate.
Tentadura, the testing of the ore from the *torta*.
Tepitate, rubbish, the waste brought out of the mines.
Tierra, land or earth, applied to the smaller particles of ore.
Torta, a cake, the heap of semi-liquid ore on the *patio*.
Tortilla, a small cake of maize-meal used as food.
Vara, the Spanish yard, a measure of 33 inches.
Veta, a vein.
Voladero, a runner, the grinding stone of the arrastra.

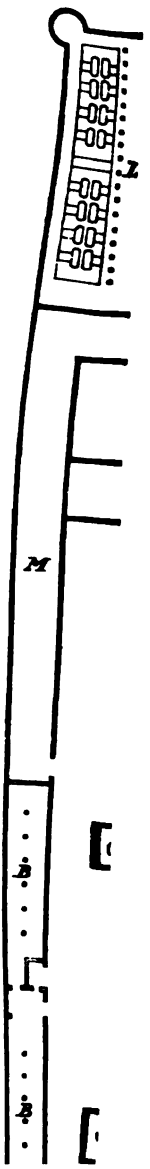
Mr. A. M. CHAMBERS moved a vote of thanks to the President for his valuable paper, and the discussion was adjourned to the next meeting.

Mr. W. E. GARFORTH seconded the motion.

The motion was carried.

ILLNESS OF MR. JOSEPH MITCHELL, J.P.

Mr. JOHN GERRARD asked whether he would be in order in moving that the President be asked to write a letter to a member now lying on a sick bed, and who had done much for the Institute as an ordinary member, in an official capacity, as Secretary, and as one of the able Presidents of the Institute? If it would be in order, he should propose that the President be instructed to write a letter expressing the deep sympathy which they all had for Mr. Joseph Mitchell in this, his trying time of sickness. He might dilate upon it, but words failed when most desired. He saw Mr. Mitchell last Saturday afternoon and knew how serious his illness was. He had seen the change that it had made in him, and therefore he spoke most feelingly. He moved that the Chairman be instructed to write a letter expressing the deep sympathy of the members with Mr. Mitchell.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Mr. A. M. CHAMBERS said perhaps they would allow him to second the resolution as an old friend of Mr. Mitchell's, feeling most deeply with him in his present hour of trial. He trusted it was not yet too late to hope some change might take place in his condition which might lead him to a better state of health, that they might again see him amongst them—at all events they all wished that most heartily.

The PRESIDENT said he was sure that they would all join with him in thanking Mr. Gerrard for bringing the subject forward. Their best sympathies were due to Mr. Mitchell in his hour of trial. He had been one of the most steadfast supporters of the Institute from its foundation; he had done everything he could for its benefit, and if they passed the resolution he should be glad to write a letter in the terms suggested.

The resolution was carried.

Mr. T. W. H. MITCHELL replied saying, "I cannot say much—thank you."

Mr. HAROLD BONSER read the following paper on "Improvements in Exploders for Shot-Firing":—

IMPROVEMENTS IN EXPLODERS FOR SHOT-FIRING.

By HAROLD BONSER.

The great advantages of using low-tension instead of high-tension fuzes, and the facility with which the former can be tested before use, are admitted. Electric shot-firing is becoming universal, the advantages of being able to safely return to a shot, even if a missed or flat shot, the absence of fume or smoke, and the precision of firing, are becoming more and more appreciated. To remove one of the greatest disadvantages, and one which has delayed the adoption of electric shot-firing, the form of exploder described in this paper has been devised.

A fireman in the discharge of his duties finds it impossible to burden himself with a bulky and heavy magneto-exploder, weighing from 8 to 10 lbs. and a bulky coil of cable, and a workman has to be specially

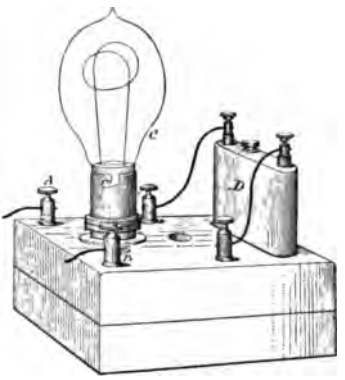

Fig. 1.

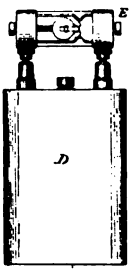

Fig. 2.

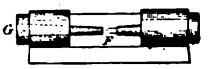

Fig. 3.

appointed, whose sole duty it is to ignite shots. With this new apparatus, the ordinary fireman may discharge this duty, as the exploder can be carried in his waistcoat pocket, and an ordinary electric light cable of 30 feet in length can be carried in another pocket.

The advantages of this apparatus are:

Portability.—The exploder can be carried in the vest pocket, weighing only about 8 to 10 ounces.

Efficiency.—The exploder shown to-day has discharged 1,000 shots, and is still ready for further use. It is desirable that the accumulator should not be allowed to run down too low, and current should be run into it from time to time. Its efficiency can be tested at any time by means of the small testing apparatus, consisting of an incandescent lamp enclosed in an air-tight tube *E* (Fig. 2), which on being applied to the terminals of the exploder or battery *D* (Fig. 2), shows by a glow that the apparatus is in a fit condition for shot-firing. The firing-cable may be tested by means of the same appliance.

Recharging.—If recharged from a dynamo, the accumulator is placed either in the charging-block *D*, shown in Fig. 1, or run in series with a 16 candle-power lamp *C*, the positive current from the dynamo being run into the terminal marked *A* and the negative wire connected with the terminal marked *B*. In some cases, to commence with, as long a time as six hours is necessary. Afterwards the current may be run in for about an hour at any convenient time. The plug should be removed when running current into the accumulator.

If no electric installation is handy for recharging at a colliery, the accumulator may be charged from a nest of Leclanché cells or dry cells. In this case no lamp need be put in the circuit. The lamp *C* is put in the circuit to check the passage of any excess of current from the dynamo. The positive and negative currents from a dynamo can be determined by applying pole-finding paper to the two terminal wires, the one which turns the paper red being the negative wire.

The safety fuze-igniter (Fig. 3) may be used by inserting the fuze into the tube *F* as far as the coil of wire *G*. On attaching the fuze-igniter to the poles of the battery, the coil becomes heated, and the powder in the fuze is ignited. The glass cover holds the sparks of the fuze until it has passed down so far into the fuze, that it may be withdrawn in safety.

Mr. LUPTON moved a vote of thanks to Mr. Bonser for his paper and for exhibiting an exceedingly interesting shot-firing apparatus.

Mr. NASH seconded the motion, which was carried unanimously.

SOUTH STAFFORDSHIRE AND EAST WORCESTERSHIRE INSTITUTE OF MINING ENGINEERS.

GENERAL MEETING,

HELD IN THE MASON COLLEGE, BIRMINGHAM, APRIL 4TH, 1895.

MR. H. C. PEAKE, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting and of the Council Meeting were read and confirmed.

The following gentlemen were elected:—

MEMBERS—

JOSEPH ARGALL, Manager, Nant-y-Mwyn Mines, Llandovery, Wales.
FREDERICK GEORGE GALE, Mine Manager, Fore Street, Camborne.
GEORGE ANDREW HURST, Colliery Manager, Dosthill, Tamworth.

ASSOCIATE MEMBER—

ANTON RÄBER, Mechanical Engineer, Aston, Birmingham.

STUDENT—

FREDERICK EDGCUMBE NICOLLS, Student, Elfordleigh, Launceston, Cornwall.

The CHAIRMAN expressed satisfaction as to the recent meeting of The Federated Institution of Mining Engineers held in Birmingham. The arrangements had been most successfully carried out; and he read a very complimentary letter received by the Secretary on the subject. He moved that the best thanks of the members be accorded to the Secretary (Mr. Alexander Smith), to Mr. Richard Chamberlain, to Messrs. Tangyes Limited, and to all who had assisted in the arrangements for the meeting.

Mr. ARTHUR SOPWITH seconded the motion, and it was unanimously approved.

DISCUSSION ON MR. E. J. BAILEY'S PAPER ON "ELECTRIC-HAULAGE BY THE MAIN-AND-TAIL ROPE SYSTEM."*

Mr. W. B. Scott said that he did not see how they could usefully discuss the paper, as further particulars were required as to the difference in cost between electric-haulage and the old system. Members were not likely to adopt electric-haulage unless they were shown that it was cheaper than the old system of haulage.

Mr. Bailey observed that he had not gone into the question of comparative costs in his paper, but would be pleased to supply the information desired.

Mr. Sopwith said that the members had at present very little information before them from a purely practical point of view. It was very necessary that they should know the cost of haulage per ton per mile. When they had all the facts in relation to cost before them, they could have a very interesting discussion.

Mr. R. S. Williamson said that he agreed with Mr. Sopwith as regards the useful qualities and efficiency of the endless-rope system of haulage, but this could not at all times be applied. The system of head-and-tail haulage, as described by Mr. Bailey, he had found to form most useful feeders to the main endless-rope. He was of opinion that if something of this kind was more generally applied, it would greatly reduce the cost of haulage. The head-and-tail rope system had done good work, and in many cases it was hard to beat if all the appliances, etc., were suitably made. He would like to know the steam-pressure used, and the useful effect of the electric plant (the latter being promised for communication at a future meeting) at the Abercanaid colliery. It might be of interest to the members if he communicated his experience as to the resistance and variable load as mentioned by Mr. Bailey. On a short head-and-tail rope electric haulage-plant, which was being used as a feeder to the main endless-rope, a water-resistance was used instead of the usual coiled-wire resistance-frames. The water was contained in a cistern, and two leaden plates were suspended by a rod, having a right and left-hand screw; when the electric motor was started, the resistance was through the water; as it was desired to increase the speed, the leaden plates were brought closer together until direct contact was caused, and the water

* *Trans. Fed. Inst.*, vol. viii., page 351.

resistance ceased. By using the water as a resistance for so short a time, steam was not produced by the current passing through the water, and the shock to the generator by throwing on the power was scarcely felt. This system of resistance had been working for some time with excellent results.*

The discussion was then adjourned.

Mr. F. G. MEACHAM exhibited samples of a poisonous vetch or pea, which had caused the deaths of horses belonging to the Hampstead Colliery Company.

The meeting then closed.

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* *Trans. Fed. Inst.*, vol. iii., page 483.

CHESTERFIELD AND MIDLAND COUNTIES INSTITUTION OF ENGINEERS.

SPECIAL GENERAL MEETING,

HELD IN UNIVERSITY COLLEGE, NOTTINGHAM, MARCH 16TH, 1895.

MR. WILLIAM SPENCER, PRESIDENT, IN THE CHAIR.

The following gentlemen were elected:—

MEMBERS—

Mr. WILLIAM BELFITT, Colliery Manager, Bretby Colliery, Burton-upon-Trent.
Mr. WILLIAM FALLOWS, Colliery Manager, Nuneaton.

ASSOCIATES—

Mr. JOSEPH BARBER, Under Manager, Ansley Hall Colliery, Atherstone.
Mr. DAVID CLAMP, Under Manager, Haunchwood Collieries, Nuneaton.
Mr. ARTHUR EDMUND HUGHES, Assistant Manager, Stanton Colliery, Burton-upon-Trent.
Mr. JAMES OLNER, Under Manager, Haunchwood Collieries, Nuneaton.

STUDENT—

Mr. MELVILLE DALYELL RADFORD RICHARDSON, Mining Student, Beech Hill, Mansfield.

The resolutions,* of which notice was given, on behalf of the Council, at and in the agenda paper for the meeting of January 26th, 1895, were moved by Mr. GEORGE LEWIS, seconded by Mr. G. E. COKE, and adopted *en bloc* without discussion.

The SECRETARY read the balloting-list prepared by the Council for the election of officers for the year 1895-96.

* *Trans. Fed. Inst.*, vol. ix., page 101.

DISCUSSION UPON MR. H. R. HEWITT'S PAPER ON "AN UNDERGROUND ENDLESS-ROPE AT THE MOSTON COLLIERY, MANCHESTER."*

Mr. G. ALFRED LEWIS (Derby) said that in a colliery with which he was connected, where the gradient was very steep, they were dissuaded from using a clip with circular jaws, owing to the injury it might do to the ropes. The gradient was not so steep as that mentioned by Mr. Hewitt, and he would be glad if Mr. Hewitt would say whether any injury had occurred to the ropes during the time that it had been working. He could quite understand the clip being of great value, because the grip on the rope was by no means perfect at the top of an incline when parallel jaws were used. It would be much better to adopt a circular jaw, if it could be used without injury to the ropes. He must take exception to Mr. Hewitt's statement that "the greatest angle of inclination of the road is 45 degs., and this angle should be taken in calculating the present working load."† He (Mr. Lewis) did not understand why that angle should be taken in calculating the working load of the rope; thus, if from any cause a small portion of the rope were to be placed in a vertical position, one might, by precisely similar reasoning, assert that the total weight of the rope and of the tubs attached to it was supported by the rope alone, which was obviously out of reason. He had worked out the tension of the rope more in detail than the writer appeared to have done. After completing the calculations with the exact inclinations as stated by Mr. Hewitt, he found that there was a great difference between his result and the one given in the paper, and that the tension of the full rope worked out to 8·4 tons. He had not allowed anything for friction, but was willing to allow 0·4 ton, whilst Mr. Hewitt allowed only 0·1 ton, therefore in that case he got the tension of the rope to be 8·8 tons. At a colliery, where the system of working was the same as that mentioned by Mr. Hewitt, he had occasion within the last few weeks to test the tension of the rope. After indicating the engines he found that on the main-belt rope running down the shaft the tension was 16 tons, and from that he calculated the tension of the hill-rope to be at least 14 tons. The 2 tons here deducted was an allowance for a small rope working a level-plane from the top of the engine-plane to the pit-bottom. The hill-rope in this

* *Trans. Fed. Inst.*, vol. viii., page 377. † *Ibid.*, page 378.

case was 1 or 1¼ inches in diameter, therefore he could not quite see that Mr. Hewitt's rope was overweighted. In his opinion a great deal more strain could be, with safety, put upon the rope, and this would increase its efficiency to a very great extent.

Mr. G. E. COKE (Nottingham) asked Mr. Hewitt how much coal could be kept in the tubs on such a steep incline? Unless the tubs were loaded with large coal only, it appeared as if a considerable quantity would fall out. In the steep mines of North Staffordshire a contrivance was used for bringing out the tubs in a horizontal position.

Mr. C. J. OLIVER (Chesterfield) asked what means were used to prevent an ascending tub from tipping up the movable platform while a tub was being attached at an intermediate landing?

Mr. A. H. STOKES (Derby) thought that the more important question was the difference between the haulage upon a steep incline by endless-rope or by main-rope. This was a far more important question than the details that were being discussed. The safe and economical mode of hauling coal up a steep roadway, whether by main-rope or endless-rope, was a subject which would give rise to a variety of opinions, and the discussion on these lines might be continued with profit. The author stated that a straight rope was taken out and an endless rope inserted with a saving of 25 per cent. of the working cost. He presumed from Mr. Hewitt's paper that at the time the main-rope was at work it was driven by a compressed-air engine, and that upon the introduction of the endless-rope haulage the motive power was changed, and a steam-engine on the surface was used; perhaps Mr. Hewitt would say whether that was correct?

Mr. HEWITT assented.

Mr. STOKES continuing, said that, therein probably was the secret of the saving. If the engine in the mine was driven by compressed-air, about one-third of the horse-power given out by the steam-engine would be utilized, but upon the compressed-air engine being removed and the power taken direct from the engine at the top by a strap-rope down the shaft, the saving would be not in the adoption of the endless-rope system over the main-rope, but by the substitution of direct steam-power for compressed air. Was it really economical to adopt the endless-rope for the main-rope? He was of opinion that a strap-rope down the shaft, driven by a steam-engine on the surface, would have done the work more cheaply than it was being done now. He had no doubt that the best system of haulage was by applying the steam-engine on the surface to drive a light rope down the shaft, and to drive hauling-ropes by gearing

from the shaft-bottom. Mr. Alfred Lewis' remarks with regard to the tension of the rope in the shaft deserved notice, but he did not think that the tension of the rope in the shaft had anything to do with the hill-rope, because if the gear were, say 3 to 1, the tension would not be the same as that of the rope on the hill. Although the pressure or load upon the rope pulling might be calculated, that was not the only resistance to be overcome, but the *vis inertia* of the rope when starting should also be taken into account. If the working load upon a rope be one-fifth of its breaking-strain when new, there would be only a very small margin of safety when starting and stopping, and especially so as the rope became worn. He should question whether it was advisable to work very steep inclines with an endless-rope instead of a main-rope haulage. If one of the clips gave way, one tub might run to the next, the next to the next, and so on, until there would be a number of tubs all together on the incline. Clips sometimes slipped and caused serious accidents. He did not say this particular kind of clip would slip, for he had never seen it, but that some do slip was well known. It was questionable whether it would not be better to draw the train straight out upon a steep incline and keep the road clear when not at work. It was a subject for careful consideration to ascertain whether the most economical and the safest way of haulage up such an incline was by means of trains or endless-ropes.

Mr. G. A. LEWIS replied, in reference to Mr. Stokes' remarks, that the speed of the two ropes in the case which he had quoted was exactly the same, and he thought Mr. Stokes would agree with him that one was a criterion of the other.

Mr. STOKES agreed that was so, if one rope ran at the same speed as the other. He was only saying it was no guide, unless the ropes were running at the same speed, and then, of course, Mr. Lewis' figures should have been 16, and not 14 tons.

Mr. J. H. W. LAVERICK (Riddings) said that he could give an instance of the supersedure of compressed air; at one of the Riddings collieries they once had a hauling-engine at the top of an inclined-plane at the end of a level stone-head, 1,200 feet from the pit-bottom, worked by compressed-air. As the incline extended, it was found that the hauling-engine was becoming rather too small for its work, and it was decided to altogether abolish the use of compressed-air. The engine, which was formerly used to compress the air, had a drum geared to it, from which a main-rope was now working down the shaft, along the level stone-head for 1,200 feet and hauling coal up the incline. There was now abundance of power, and the new had proved a great improvement on the old system.

Mr. HENRY LEWIS asked Mr. Laverick what tonnage he was hauling per day?

Mr. LAVERICK: About 200 tons along the old plane, and about 200 tons more along a new plane by the same rope.

Mr. HENRY LEWIS asked Mr. Hewitt what provision was made to prevent tubs from running down the plane and causing great damage? He could not believe that back-stops would be of any use on so steep an incline.

Mr. H. R. HEWITT (Derby) was very pleased to see that his paper had brought forward so interesting a discussion. Figs. 8 and 9 showed the

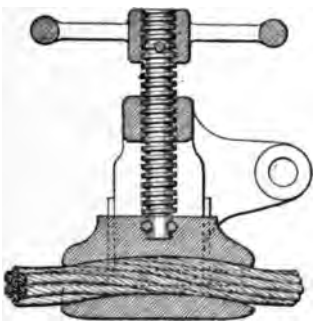


Fig. 8.

Fig. 9.

clip as used at Moston colliery. He thought he was right in saying that no injury had been noticed on the rope due to the working of these clips, as the jaw of the clip was formed to a radius of 3½ feet. He quite agreed that if a shorter clip had been used much damage might have been caused to the rope. Other clips had been tried on the incline, and had been found to be unsuitable for use, so there was no other course to adopt but to design a clip to suit the particular purpose. He had expected criticism upon his calculations in which he had taken the greatest inclination of the road as a basis, and the resultant working-load would be the same as that given by his figures if the road had a regular dip of 45 degrees from the top to the bottom. He had thought it advisable to show the amount of work that this rope would be doing under the very worst conditions which could possibly occur on this road. The average weight of coal contained in a tub when it reached the top of the incline was 10 cwts. No slack was sent out of the pit, and the tubs were well packed. Bells were fixed at each landing connected to levers placed at points 30 feet up and down the plane. These levers were arranged in such a way that a passing tub rang the bell, and thus gave audible warning of its approach. Mr. Stokes was right in assuming that compressed-air was used for working this

haulage previous to the present steam-engine being placed on the surface. There were several inclined-planes similar to the one described in his paper, and a compressed-air engine was used at the top of each one driving a single rope, but these were gradually being replaced by the endless-rope system described in his paper. The loosening of the clips and the running back of the tubs mentioned by Mr. Stokes and Mr. Henry Lewis was a very serious matter. Members would quite understand that no stop-blocks would be of any service whatever in a road of this inclination, and if the tubs did run back, great damage resulted. Previous to the clip described being used, hitching-chains were put on behind each tub. These consisted of a short length of chain, which was coiled twice round the rope, and both ends hooked on to the drawbar at the back end of the tub. When the clip failed to act, the tub immediately ran back a few inches until the chain was tightened, and the tub then completed its journey on this hitching-chain. If suitable precautions had not been taken, the pulley at the brow of the hill would have cut the chains in two, thus liberating the tubs. Consequently, a boy was stationed there to stop the rope if he saw a tub being brought up on the chain, and his duty was to tighten the clip or to use another and remove the hitching-chain. It was found by using the clip, which had been described in his paper, that no hitching-chain was required, and the clip was working in a very satisfactory manner. It was very necessary that the clips should undergo a daily examination, as if a bad clip were used the tubs would be liberated from the rope, and cause considerable damage. This clip had been in use for four years, and accidents due to the failure of the clips had been of very rare occurrence, owing, firstly, to their design, and, secondly, to their being kept in good order.

Mr. HENRY LEWIS asked whether in putting the clip on the rope the rope had to be stationary? And what number of tons was being hauled up one of these planes?

Mr. HEWITT replied that the rope was running during the whole working shift, and was never stopped from any cause whatever. The quantity of coal hauled on one incline was 300 tons per day.

The PRESIDENT regretted that he had not been able to read Mr. Hewitt's paper owing to illness. He approved the suggestion that consideration of this paper should be deferred, and no doubt more valuable information would be brought forward to elucidate the important points Mr. Stokes had mentioned, and to compare this method with that of having the rope taken in by the empty tubs which, with such a gradient, had generally been the course adopted hitherto in this district. By the

use of the endless-rope, the weight of the empty tubs going down helped those coming up and to set against this was the possible damage to the rope by a screw-clip on a heavy gradient, which, however, with only one tub at a time, might not be serious

The discussion was then adjourned.

Mr. WILLIAM HAY (Stanton Colliery, Burton-on-Trent) read the following paper on "Banking-out, Screening, and Mechanical Arrangements for Direct-sorting and Packing of Coal" :—

BANKING-OUT, SCREENING, AND MECHANICAL ARRANGEMENTS FOR DIRECT-SORTING AND PACKING OF COAL.

By WILLIAM HAY.

The banking-out of coal when it reaches the surface and its subsequent treatment by screening, cleaning, etc., have not until quite recently received in this district that amount of attention which their importance demands. In these days of keen competition, the preparation of coal for the market by separating the smaller class from the larger, and the removal of foreign substances from all qualities of coal, are matters of great moment, and will, if economically and efficiently accomplished, largely contribute to the commercial prosperity of a colliery.

Wherever steam-power is available and can be utilized (without the introduction of too complex machinery) in the assisting of or the dispensing with manual labour, its adoption will invariably be profitable; and at no place can it be more advantageously employed than in connexion with the surface arrangements of a colliery.

Before the present plant at the Stanton colliery was erected, the system in use was the one commonly adopted in the Midland counties of hand-sorting from the boxes or tubs. The tubs after having been weighed, passed to the trucks standing by the side of the bank. Here the workmen selected the class of coal from the tubs with which their waggon had to be loaded, and placed it upon the side-board affixed to the waggon. The men standing in the truck then put it into position in the waggon. In this case all large coal had to be twice handled before being finally placed. The portion of the coal remaining in the tub was then hoisted up to an altitude of 15 feet and tipped over fixed screens. By this method all the large coal remaining in the box passed directly into the truck, and not being hand-placed was sold as seconds coal.

Banking-out.—It may be mentioned at the outset that at the Stanton colliery coal is drawn out of two shafts, about 30 feet apart, by one winding-engine from two different depths of 600 and 750 feet respectively. The shafts being only 7 feet in diameter are, of course, too small to admit of two cages being placed in one shaft. When the cage lands at the surface and rests on the props, the full tubs are taken off by hand and the empties put into the cage on both decks simultaneously (Fig. 1, Plate XI.).

The timbers to which the props are attached are fixed on telescopic cast-iron boxes *A* containing springs, 9 inches long and 5½ inches in external diameter (Fig. 2, Plate XI.) When the load is off the props the spring-boxes are 2½ inches from being closed. In the event of a load being suddenly dropped upon them the telescopic boxes would come together if depressed 2½ inches, and thus prevent permanent damage from being done to the springs. The ordinary working-load of nearly 4 tons causes a depression of about ¾ inch. It will be observed that any lateral movement of the prop-timbers is prevented by the strong cast-iron brackets which act as guides to the timbers, the latter being plated opposite the brackets, and kept well lubricated to admit of an easy up-and-down movement. These springs are invaluable, they add considerably to the durability of the cages, conduce to the safety of the winding-ropes, and assist the banksmen in the unloading of the cages by gently rising as the loaded tub is being pushed out.

When the cage is on the props it forms a gradient of 1 in 70, and the full tubs run at a like gradient to the full gravity or drop-cage. The empty tubs are hoisted at the same time on the opposite side of the pits, and run to the respective shafts on gradients of about 1 in 50. (Fig. 2.) When the full tubs have been weighed they pass on favourable inclinations to the tippler, and after tipping run out to the creeper-chain, which elevates them up to a point whence liberated they run gently to the turntable (Fig. 1.)

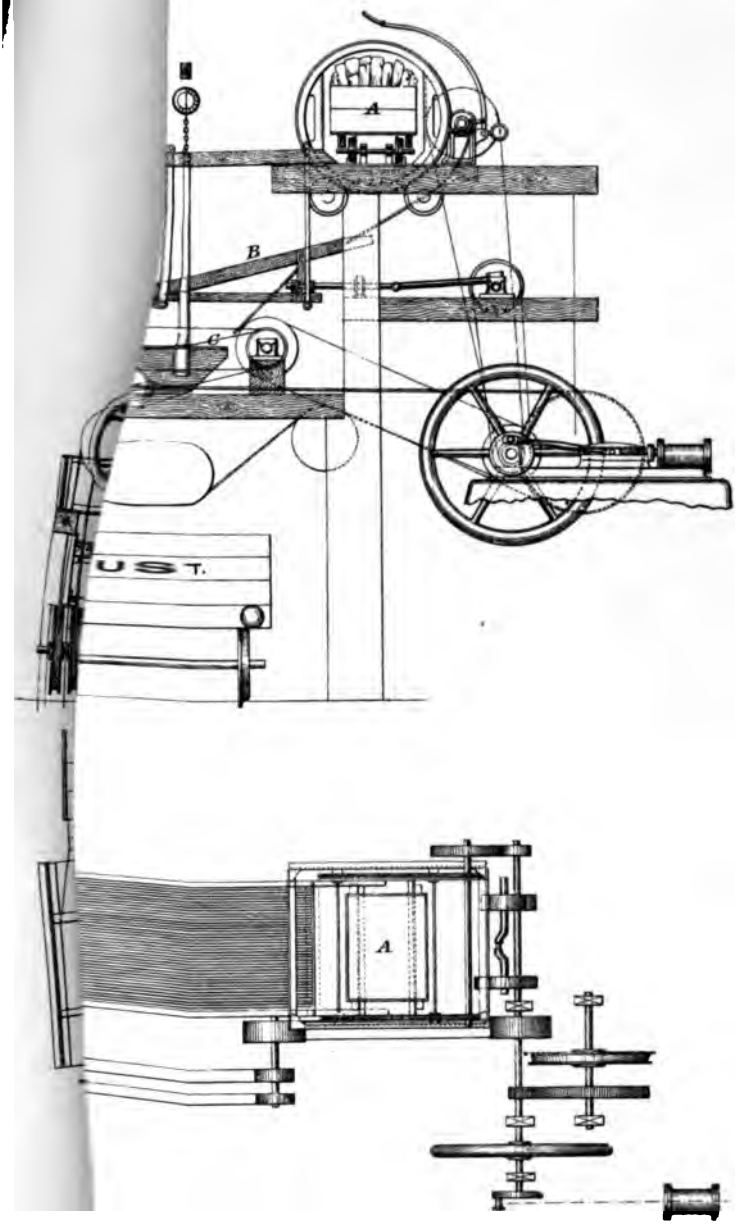
Tippler.—A side-tippler *A* is used, worked by an ordinary conical clutch. One of the advantages possessed by this tippler over other forms of side-tipplers in vogue, is that the tubs when emptied can pass out at the opposite side, and much time is saved. It is capable of tipping 5 tubs in 2 minutes, causing very little breakage of coal, there being a fall of only 9 inches. The tub when being tipped is held in position by four claws—two on each axle. The workman in charge of the tippler can, when necessary, change the speed of rotation by varying the tension applied on the handle (Figs. 3 and 4, Plate XII.).

Screens, etc.—The coal is tipped on to a jigging-screen *B*, having ordinary screen-bars 8 feet long, placed ¾ inch apart, to allow the dust to be taken out. The dust passes directly into a billy-fairplay weigh-box *C*, and the quantity of dust loaded by the workmen into each tub is readily ascertained, and registered in a book. Motion is imparted to this jigging-screen by a connecting-rod coupled to a bell-crank, whose shaft is driven by a belt-pulley running at 75 revolutions per minute. It has a reciprocating motion of 7 inches. It is suspended by means of vertical iron rods attached to the carrying timbers.

The coal passes directly from this to another jigging-screen *D*, 13½ feet long by 5 feet wide (having two sets of interchangeable screen-bars), and running at 95 revolutions per minute. The upper set of screen-bars covers a surface of 30 square feet with 2¼ inches openings, and the lower set has an area of 22½ square feet, with 4 inches openings, through which the kibble-coal pass. This jigging-screen has an inclination of 1 in 6, and is suspended by means of wooden rods reduced in thickness in the middle, which afford an easy motion to the screen. Immediately underneath the upper screen-bars is fixed a wire-gauze of 1⅛ inches mesh, on to which the nuts are passed. Any dust which may be made in transit to this point is here removed. This dust is conveyed by a shoot into the dust-waggon, and the nuts pass forward on to a conveying-jigger *E*, placed at right-angles to the two preceding jigging-screens. As the coal is friable, it falls on to boards, instead of an iron-plate, and this is found to save much breakage of coal. Boys are stationed by the side of the conveying-jigger *E*, who clean the coal before it passes into the trucks. Before the introduction of this nut-cleaning jigger, a 2 feet gandy-belt was used, but considerably better results are now obtained. After repeated trials and tests of the waggons loaded, generally a few pounds of stones and from 4 to 5 per cent. of dust are found in a 7 tons truck of nut-coals, thus showing that the coals are fairly well cleaned. About 30 to 35 cwts. of small stones and batts are picked off per day of nine hours.

The kibble-coal is conveyed into the truck by means of a movable pan *F*, worked by levers *G* (attached to the ends of the jigging-screen *D*), pivoted somewhat out of the centre, so as to reduce the travel from 7 to 4 inches. The end of the pan *F* is supported on a wooden roller. The coal is lowered into the waggon by a balanced shoot *H* by means of a brake.

Cleaning-belt.—The large coal remaining is pushed or propelled on to a belt *I*, 3½ feet wide and 30 feet long. Workmen are stationed on this



8 FEET TO 1 INCH.

It is carried on angle-irons bolted to cast-iron standards on each side. No rollers are used of any description, the links being supported by and carried on wrought-iron joists which admit of no bending of the belt when loaded, and consequently, in the writer's opinion, adding very considerably to the life of the belt. This belt travels at the speed of 37 feet per minute.

It may be added that the man in charge of the tippler has control over the engine by a suitable balanced lever, and there is an electric bell fixed close by him, from which connexions are distributed over the whole system, and by means of which the machinery can be stopped immediately.

Motive Power.—The power working the whole of the screening and other arrangements is obtained from an engine having a cylinder 10 inches in diameter and 18 inches stroke, running at 100 revolutions per minute, and developing 17 indicated horse-power, with a mean steam-pressure of 25 pounds per square inch. The rope-pulley driving the two belts is placed on a countershaft, and is geared about 7 to 1.

Rope.—A rope $\frac{3}{4}$ inch in diameter is used to drive the belts: its life is somewhat short (about six months), owing to the numerous twists and turns on the various pulleys over which it passes. An endless chain is being tried, and appears to give, so far, satisfactory results, but has not been in use a sufficient time to justify a definite opinion. The first cost of the rope is small, as it is only 480 feet long. The slack rope is taken up by means of a balance or weighted pulley.

The percentages of the different classes of coal made are :—

Hand-placed coal	48 per cent.
Kibble coal	11 „ „
Nut coal	20 „ „
Dust coal	17 „ „
Dressings	4 „ „
	100

The wage cost for a quantity of 700 tons per day, and the average cost per ton, are as follows :—

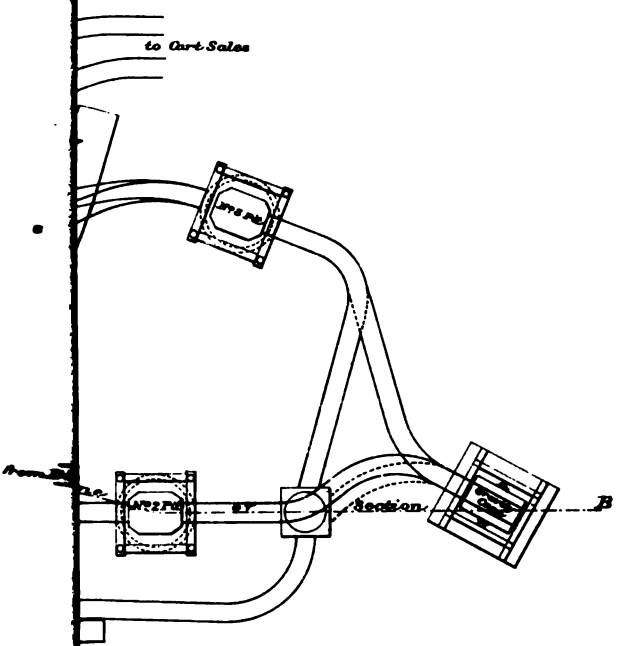
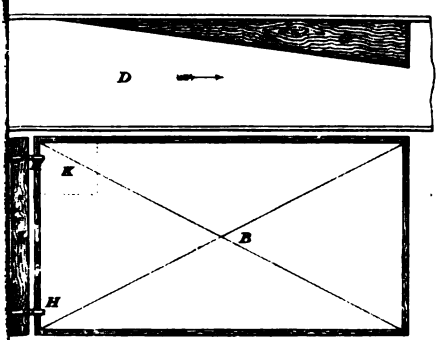


FIG. 5.



PLAN.

SCALE 4 FEET TO 1 INCH.

Class of Workmen.	s.	d.		Amount. s.	d.	Total Amounts. £	s.	d.		Average Cost per Ton. d.
Banking and Tipping—										
1 Principal banksman, at	4	3	...	4	3					
1 Banker-out, at	3	6	...	3	6					
2 Assistants, at	3	0	...	6	0					
2 Catch-boys, at	1	6	...	3	0					
1 Greaser, at	1	6	...	1	6					
2 Shunters, at	3	3	...	6	6					
1 Turntable-boy, at ...	2	6	...	2	6					
1 Tipper, at	3	3	...	3	3					
1 Billy-fairplay-boy ...	2	0	...	2	0					
						1	12	6	...	0·55
Picking small coal—										
2 Men, at	3	3	...	6	6					
2 Youths, at	2	3	...	4	6					
1 Boy, at	1	6	...	1	6					
						0	12	6	...	0·21
Belts—										
4 Cleaners, at	3	3	...	13	0					
5 Loaders, at	3	3	...	16	3					
5 Loaders, at	2	3	...	11	3					
						2	0	6	...	0·70
General labour—										
3 Shunters and cleaners, at	3	3	...	9	9					
1 Oil-boy, at	2	0	...	2	0					
1 Foreman, at	4	3	...	4	3					
						0	16	0	...	0·28
Totals						5	1	6	...	1·74

The PRESIDENT said that the discussion would be adjourned. He had great pleasure in proposing a vote of thanks to Mr. W. Hay for his important paper.

The vote of thanks was unanimously passed.

DISCUSSION UPON MR. M. DEACON'S PAPER ON "THE USE OF EXPANSION-GEAR AS APPLIED TO COLLIERY ENGINES."*

Mr. J. H. W. LAVERICK (Riddings) asked Mr. Deacon how the expansion-gear affected the running of the engines, whether there was any increase or otherwise in the number of "draws" in the hour, and what was the saving in the consumption of fuel?

Mr. C. J. OLIVER (Chesterfield) asked whether Mr. Deacon could give the speeds of winding when the diagrams reproduced in Figs. 1, 2, 3, and 4 (Plate XIX.), were taken, as these appeared to show a considerable reduction in speed in the cases of Figs. 2 and 4. Could he also give the speeds at which the other diagrams were taken, especially the fan-engine diagrams (Figs. 5 and 6)? In the case of the Guinotte expansion-gear, he noticed that there was a saving of 9 per cent. of steam, but a loss of about 9 per cent. in time in winding, which would mean in the case of a pit winding 1,000 tons per day, a loss of 100 tons in the daily output—an amount which would probably far more than counterbalance the saving in steam, especially where the utmost weight of coal had to be wound in the day. It was said in reference to the Melling slide-valve expansion-gear that "When steam is shut off at the throttle-valve and the speed of the engine falls to that at which the gear commenced to act, it ceases to act, and the engineman is free to admit steam for reversing as if the cut-off gear were not attached."† He asked whether the engineman had no control over the expansion-gear while the engine was in action at full speed, that is to say, whether he could admit steam against the engine in order to stop it, in case the throttle-valve could not be closed promptly?

The PRESIDENT said that the use of expansion-gear was an extremely interesting subject, and he regretted that he had not yet been able to give it the attention which it deserved. A very important saving of steam, about 25 per cent., was stated as having been obtained. There was nothing more desirable to be saved than steam, as there was an element of danger in its use. It had been prognosticated by an eminent engineer

* *Trans. Fed. Inst.*, vol. vii., page 672; discussion, vol. viii., page 373.
† *Ibid.*, page 680.

that steam-engines would cease to be used during the lifetime of some members present that day, but we seemed to be a long way off that yet.

Mr. DEACON, replying, said he was glad to find that Prof. Lupton agreed with the main points in his paper, especially in the necessity of high-pressure steam being used so as to attain substantial advantages from expansion in non-condensing engines. His statement that it was necessary to have about 20 lbs. of absolute pressure in order to overcome the opposing pressure of the atmosphere and of the friction of an ordinary colliery engine was correct. It might perhaps not be out of place to point out that absolute pressure was calculated from zero, which was 15 lbs. below the atmospheric line when the barometer indicated a pressure of 15 lbs. per square inch. Thus, 20 lbs absolute, meant 5 lbs. of indicated steam-pressure in the cylinder. He thoroughly agreed with the purport of Mr. Lupton's remarks on that point, which went to show that the terminal absolute pressure should not, to attain the best results, be less than sufficient to balance the friction of the engine and the opposing atmosphere, and probably a slightly higher pressure would be better. If a very low terminal pressure were used, a greater degree of loss from condensation and re-evaporation was caused by the reduced temperature of the expanded steam robbing the walls of the cylinders of some of the heat previously imparted to them by the high-pressure steam before expansion commenced, producing a certain degree of cooling of the cylinders which caused partial condensation of the fresh steam admitted at the commencement of the return stroke; on the other hand, better results were effected owing to a reduced back pressure. Prof. Lupton appeared to consider that an earlier cut-off than one-half should not be used with an initial pressure of 60 lbs. per square inch in the cylinder. This initial pressure would leave an absolute terminal pressure of 37½ lbs., or 22½ lbs. above the atmospheric line. This pressure was in his (Mr. Deacon's) opinion unnecessarily high, and not calculated to produce the best results. This opinion was confirmed by the results of some carefully conducted experiments made by Messrs. Musgrave & Sons, to ascertain the varying weights of water evaporated per indicated horse-power by the same engine with varying degrees of cut-off. The engine experimented upon had a cylinder 16 inches in diameter and was non-condensing. A large number of tests were made at initial pressures varying from 40 lbs. to 100 lbs. per square inch, but, for the purpose of comparison with the figures mentioned by Mr. Lupton, viz., 60 lbs. initial pressure and 50 per cent. cut-off, the results of the tests made at 60 lbs. initial pressure only have been taken, as follows :—

Cut-off at per Cent. of Stroke.	Initial Pressure in Cylinder.	Mean Effective Pressure.	Feed-water required per Indicated Horse-power per Hour.	
			Corresponding to Typical Diagrams with no Leakage.	Corresponding to Actual Result from Practical Tests. Slight Leakage.
	Lbs. per Sq. In.	Lbs. per Sq. In.	Lbs.	Lbs.
10	60	8·70	37·26	40·95
20	60	21·12	27·55	29·43
30	60	30·47	27·24	29·10
40	60	37·75	27·92	29·63
50	60	43·42	28·94	30·66

If these results were correct, it appeared that greater economy arose with the cut-off at 20 per cent. than at 50 per cent., the best results being at 30 per cent., although only very slightly better than 20 per cent. With initial pressures of 80 lbs. and 100 lbs. per square inch the best results were obtained at 20 per cent. cut-off (25·68 lbs. and 23·77 lbs. of water evaporated respectively in the actual tests), and the worst at 50 per cent. cut-off (28·38 lbs. and 26·99 lbs. respectively). In the indicator diagram, Fig. 22 (Plate XX.), the initial pressure was 75 lbs., and the boiler-pressure being from 90 to 100 lbs. per square inch, a very much earlier cut-off than one-half was economically applied to the engines in question, where the actual saving in fuel ascertained by careful experiment was about 25 per cent. The indicator diagrams exhibited by Mr. Davies (Figs. 1 and 2*) were good illustrations of what might be done by careful attention to the proper proportioning of the valves and steam-passages to the area of cylinder; by the proper regulation of the valve-gear; and by the use of expansion-gear. Mr. Davies questioned the invariable accuracy of Mr. Deacon's statement that "a decreased speed of running is the inevitable result of the use of expansion-gear." This statement was made on the assumption that the valves and steam-passages were properly proportioned to the size of the cylinders, that they were the same with or without expansion-gear, and that the engine before having the expansion-gear applied was doing good work. Of course, if through the undue smallness of the valves and contracted area of the ports and passages the engines were doing bad work owing to wire-drawing and back pressure, an acceleration of speed instead of a diminution might be attained by increasing the size of the valves and steam-passages, even with the adoption of expansion-gear. In the case of an average winding-engine, however, with steam on from 0·875 to full stroke, and giving an average

* *Trans. Fed. Inst.*, vol. viii., page 374.

indicator-diagram, it would generally be found that the speed of winding would be slightly decreased by the use of expansion-gear if an early cut-off were used, but this might be rectified by increased boiler-pressure or by increasing the size of valves and steam-passages. If an engine was working to its full power, with steam on throughout the stroke, there would be a slightly decreased speed of winding with expansion-gear unless the initial pressure in the cylinder were increased either by additional boiler-pressure or by facilitating the free passage of steam into the cylinder by large valves and steam-passages. If Mr. Laverick referred to the paper he would find a comparison of the speed of winding with and without expansion-gear and the saving in fuel. He had not particulars of the speeds of winding in the case of the indicator-diagrams shown in Figs. 1 and 4; but the diagrams being from different engines a comparison of speeds could not be very useful. Fig. 1 was taken from an engine with large valves and ports, whilst Fig. 4 was taken from an engine with a cylinder 36 inches in diameter, with valves and ports which were designed for an engine with a cylinder 30 inches in diameter (the 30 inches cylinder having been replaced by the 36 inches cylinder without any corresponding increase in the size of valves, etc.). Fig. 4 was interesting as being a pretty diagram on paper, but it represented excessive wire-drawing and consequent loss of economy through restriction in the steam-passages. The fan-engine diagrams, Figs. 5 and 6, were both taken at the speed of 46 revolutions per minute. In the case of the engine fitted with the Guinotte gear, the reduced speed of winding resulted in no loss of output, as the engine was capable of raising all the coal required in a given time with the gear in use. If the engine had been too small for its work, the gear could not, of course, have been usefully employed unless additional steam-pressure had been available. There was nothing to prevent reversing of the Melling gear when it was in action. In conclusion, he said it must remain with each individual to decide as to whether the use of expansion-gear would or would not be economical in any particular case, but he hoped that his paper had thrown some light upon the subject, which might be of use to members in deciding so important a question.

The PRESIDENT said that the members were all very much indebted to Mr. Deacon for his paper and the important reply that he had just made, but it did not seem to have excited that interest amongst the

members which he should have thought it would have done. He had great pleasure in proposing a vote of thanks to Mr. Deacon for his interesting paper, and that the discussion be closed.

The vote was accorded with acclamation.

Mr. L. W. DE GRAVE read the following paper on "Continuous-current Dynamos and Motors":—

CONTINUOUS CURRENT DYNAMOS AND MOTORS.

By LANCELOT W. DE GRAVE.

There are several points connected with the working of electric lighting and power plants, more especially in works and mines, which have received little or no attention, although in everyday practice they are of vital importance. The causes and remedies of the minor troubles, to which even the best constructed machines are subject, may be the cause of a partial or total stoppage of work and consequent loss and expense. The author wishes more especially to deal with these troubles, their prevention and remedies. A brief description of the three forms of continuous-current machines may be of use to those not intimately acquainted with electrical machinery.

Continuous-current machines are designated by the manner in which the magnets are wound: as series-wound, shunt-wound, and compound-wound.

In a series-wound machine, the whole of the current generated passes through the magnet-winding; the magnetism and therefore the voltage depending on the work the machine is doing, if the speed be kept constant.

In a shunt-wound machine, a small part of the current is sent round the magnet-windings, which are of high resistance and form an independent circuit. In this case the magnetism and therefore the voltage is practically the same, whatever work the machine is doing, if the speed be kept constant. As a matter of fact, to obtain constant voltage, it is necessary to slightly increase the speed as the load increases.

A compound-wound machine is a combination of shunt and series-winding, comparatively few turns of series-winding being used, through which the main current passes, and by so doing increases the magnetism as the main current increases, thus keeping up the voltage without it being necessary to increase the speed.

Series-wound dynamos are used for arc-lighting in series. This method has no advantage except in cases where a large number of lamps, extending over a considerable area, are required: a condition which is

but rarely found except in lighting streets, railway-stations, or sidings. Series-wound dynamos are unsuitable for incandescent lighting; but for power work, where only one motor has to be driven, a series-wound dynamo and a series-wound motor form a perfect self-regulating arrangement. Series-wound motors which have a maximum torque at starting, are eminently suitable for driving machinery, where there is a heavy load at starting—for instance in pumping and hauling.

Compound-wound dynamos are the best for incandescent-lighting and arc-lighting over small areas, on the same mains as incandescent lamps and for power work where more than one motor, either series, shunt, or compound-wound, is to be driven from the same dynamo. For transmitting less than 10 horse-power over moderate distances, a compound-wound dynamo, with a voltage of say 110, may be conveniently employed for both lighting and power; but for greater horse-powers it is essential that the pressure be increased to 300, 500 volts, or more: 500 volts being the maximum pressure advisable for use underground, otherwise either the cost of cables becomes heavy, or the loss of power in the cables brings about a most uneconomical state of working. The most economical arrangement is obtained by having a definite relation between the size of the cable (and therefore the cost) and power lost in it. In cases where lighting is altogether subsidiary to power, lamps can be connected in a series of 2, 3, 4, 5, or more across the mains (the number being dependent upon the voltage), if, however, one lamp be burnt out, the whole series is extinguished until a fresh lamp is inserted. In cases where any considerable amount of lighting is required, it is advisable to have a separate dynamo running at about 100 or 110 volts.

Shunt-wound dynamos can be used for all purposes for which a compound-wound machine is suitable; the latter having the advantage of automatically maintaining a constant voltage at all loads, if the speed be kept constant. Shunt or compound-wound motors are suitable for work in which the load is small at starting, as in the case of fans, etc., and these forms of winding give nearly constant speeds at all loads.

The author desires to emphasize the necessity of providing ample motor-power: no motor should be put to work without a margin of at least 25 per cent. above the calculated power. The importance of this point is clearly shown in the paper which Mr. Alexander Siemens read before the North of England Institute of Mining and Mechanical Engineers.*

* *Trans. Fed. Inst.*, vol. viii., page 247.

Unfortunately price is a strong factor in deciding the size of a plant, and there is a natural tendency to keep down the power of the machine, a false economy.

In designing an electrical plant, it is desirable to remember that machinery is more efficient the nearer it is kept at its full load. It is therefore frequently necessary to consider the advisability of putting in two or more small machines in place of a large one, and making suitable arrangements for connecting them together when required. Such an arrangement has the additional advantage of simplifying the number of spare parts required.

Dynamos and motors should be fixed in dry places, free from dust, and as cool as possible. These conditions ought to be easily obtained for a dynamo, but are not always attainable in placing a motor. Where placed in a damp position the motor should be kept at work as much as possible; in a dusty position, the motor should be partitioned off, and with attention and the judicious use of a stiff brush, all trouble may be avoided. In such cases, one of the forms of enclosed motors (although not quite so easy to handle as the usual form) might be adopted with advantage.

The author has had experience of a case in which two 10 horse-power compound-wound dynamos were stored in a very wet shed, with a leaky roof, for nearly five years. On being asked whether they were in a serviceable condition, he naturally had some doubts on the subject; however, after they had been very carefully dried, cleaned, and kept moving slowly for nearly a week, they were put to work, and have since run for more than a year without any signs of trouble.

The two methods of driving are (1) direct-coupling, or (2) belt or ropes. Direct-coupled engines and dynamos are now becoming very popular, the advantages including the use of slow-speed dynamos, high efficiencies, and small floor-space.

Two direct-coupled plants, each of 50 horse-power, recently erected by the author for his firm, Messrs. John Davis & Son, occupy a floor-space of about 8 by 4 feet each, and run at a speed of about 450 revolutions per minute. Direct-coupled plant are not, however, recommended for steam-pressures below 80 lbs. per square inch, or in cases where the load is very variable, as in power-transmission, in which at one moment the plant may be running light and a minute later at full load. For lighting purposes direct-coupled plant forms the best arrangement, more especially when it is kept at or near its full load, additional machines being started up and coupled in as the load increases.

Belt-driving is suitable for machines up to, say, about 40 horse-power, above which ropes give the best results. For the satisfactory working of belt or rope-driven plants careful attention should be given to the length of drive and size of pulleys, but no definite rules can be given. For belt-driving nothing can equal the leather link-belt. There are cases of belt-driving in the ratio of 5 to 1 where the slack side (which should always be on the top) is within 6 inches of the bottom without signs of slipping. This fact is an advantage in power transmission, as when a heavy load is suddenly thrown on the dynamo the belt will slip, and thus takes the strain off both the engine and dynamo. In rope-driving the best results can only be obtained by using wheels with grooves carefully designed.

All foundations should be, preferably, of concrete, although bricks set with cement (having either a stone or timber top) are frequently employed. For large foundations the bottom of the concrete should be slightly arched. This will prevent the foundation from rocking should the ground at the centre be rather harder than at the edges.

Belt or rope-driven dynamos are, or should be, provided with slide rails and tightening screws, so as to take up the slack of the belt. The bed should be made quite level, the armature-shaft should be set parallel to the driving-shaft, and any slight adjustment required so as to place the belt in the line of the centre of the dynamo pulley, can be effected by means of the tightening screws.

The writer prefers, when driving machinery from motors, to use belts or ropes when ample space is available, as he considers that high-speed pinions are at the best a source of annoyance and noise.

In dynamos and motors of reliable make and modern design, faults should be few. In the best machines, however, left as they frequently are in the hands of men unaccustomed to their use, small faults are liable to occur which an experienced man could easily prevent or promptly stop, but which, if left for some time without attention, may necessitate stoppage for repairs. With the exception of the lubrication of the bearings, practically the only attention required by continuous-current dynamos and motors is at the commutator and brushes. Brushes are made of many materials, brass or copper gauze, carbon, sheet-copper, phosphor-bronze, brass wires, etc. Brass gauze on a copper commutator, or a gauze impregnated with a heavy mineral oil and black-lead make perhaps the best form of brush, the latter giving exceedingly good results.

Sparking at the brushes is the most general fault, and, if not attended to, the most destructive trouble found in dynamos and motors.

The prevention of sparks (with modern machines) rests entirely with the attendant. Machines which run without sparking at no load will spark as the load increases, if the lead (that is to say the position of the brushes on the commutator) is not altered. In dynamos, the brushes should be moved forward, that is in the direction of rotation, as the load increases, and in the case of motors should be moved backwards, the reverse movement being necessary as the load goes off. Accumulations of dust and grit, at the commutator or brushes, will cause sparking. The commutator should be wiped (about once a day unless situated in a very dusty place) with a clean cotton rag, just moistened with mineral oil (vegetable oil must never be used, and oil should never be poured on the commutator), paraffin or vaseline is the best lubricant. The brushes should also be wiped occasionally, where they bear on the commutator. If the brushes have worn away, sparking will occur; they should then be filed to the proper bevel and clipped at the tips until about $\frac{1}{16}$ inch thick, care should also be taken to ascertain that they bear in the proper position on the commutator. In two-pole machines, the brushes should be placed diametrically opposite each other, and in four-pole machines, they should be placed at right angles. All brushes should project the same distance from the holders. The brushes should press firmly but not heavily on the commutator; if they press too lightly, they will be liable to vibration, and if they press too heavily they will plough the commutator; in either case, sparking will be produced, necessitating the turning or filing of the commutator, entailing a stoppage of work, and if repeated often a new commutator.

A trouble technically known as a "flat" sometimes occurs on commutators without any apparent reason. By a flat is meant the burning away of one or more of the commutator-bars in patches, and this, if not attended to, may spread to several of the adjacent ones. A flat is generally caused by vibration, making the brushes jump; it may be cured by putting extra pressure on the brushes. More often it will be found that the jumping is due to the running of a belt with a bad joint, the vibration being caused by the joint striking the pulley; the remedy is of course to use an endless belt. Bad foundation may also cause vibration. An imperfectly soldered joint between the commutator-bars and the armature, a short circuit between two commutator-bars, owing either to the presence of copper-dust between the connexions to the armature or from a burr on the commutator will produce a similar result, and in each case the remedy is obvious. Many makers place a vulcanite disc over the connexions from the commutator to the armature, to prevent the accumulation of copper-dust: this is frequently a death-

trap for the commutator. If a collection of dust is suspected, it is better to remove the disc altogether and trust to the occasional use of a stiff brush to prevent the trouble. When a flat occurs, it is desirable, at the first opportunity, to turn the commutator down in a lathe or to file it whilst in position (using plenty of chalk on the file). When the fault occurs, it goes from bad to worse, and the longer it is left, the more the commutator has to be turned down. Turning in a lathe is better than filing, as it is difficult to make the commutator truly cylindrical with a file. After turning, the commutators should be polished with coarse and fine emery-cloth, and then examined to see that there are no particles or burrs between the bars or connexions.

If the engine or source of power from which the dynamo is driven has a tendency to run backwards either at starting or stopping, the brushes should not be put on until the engine has started, and should be removed just before the engine stops.

If a shunt or compound-wound dynamo or motor fails to excite at starting, it is generally due to a broken, dirty, or loose connexion at the brushes or terminals of the shunt-windings. When the brushes are a long way out of their proper position on the commutator, it may account for the machine not exciting; search should be made for a faulty connexion, and the brushes should be rocked as far as they will go both ways. If a battery and galvanometer are available, tests can easily be made to see if there is a broken wire. Occasionally it is found that the magnetism has been reversed, and that the machine fails to excite. In this case it is necessary to change the brush-connexions round. Shunt-wound dynamos excite best when the main circuit is open, and series-wound dynamos will only excite on a closed circuit. When a series-wound machine fails to excite, it is due to there being too high a resistance in the main circuit, and this may be caused by a bad or broken connexion in the series-winding mains or at the brushes. If no faults of this latter sort can be found, the terminals of the machine should be momentarily short-circuited by means of a piece of wire, and this test will show whether the fault is in the machine or in the outside circuit.

Short-circuiting may possibly occur in the shunt or series-coils; the short-circuited part will be cold and the remainder excessively hot.

The burning out of an armature either in a dynamo or motor is generally due to overloading, the excessive current burning or charring the insulation, and so putting the conductor in connexion with the frame-work or other conductors, furious sparks invariably ensue. The armature then requires re-winding, and should be returned to the makers.

Starting-switches, that is to say, switches combined with resistances, should always be used for starting motors. For shunt or compound-wound motors, the resistance should be inserted in the armature-circuit, and cut out as the speed increases. For series-wound motors the resistance should be placed in the main circuit. In all cases the resistance should be arranged so that not more than half of the current can pass when switched on, the resistance being gradually cut out as the speed increases. A commutator which has received proper and careful attention should soon acquire a black burnish (this is the criterion of good running), but sparking, even if permitted for a short time, will destroy the burnish and cut up the commutator.

To ensure good running of electrical machinery, cleanliness is essential, and oil and copper-dust should never be allowed to accumulate. The writer has seen machines covered with oil and dust, the invariable consequence being breakdowns or a serious shortening of the life of the plant. This point cannot be too often impressed upon attendants, as neglect is a frequent source of trouble, and one for which there is no excuse and only one remedy.

Mr. A. H. Stokes said that Mr. de Grave had written a paper containing a number of practical suggestions which would be useful to those using or thinking of erecting electrical machinery. In the first place the author said that in erecting a motor, a margin of at least 25 per cent. above the calculated horse-power required should be provided. Some engineers had found out this axiom at great cost, as their electric current was defective, their motor was bad, and they expressed their intention of reverting to steam-engines, simply because they had not provided a motor that was large enough for the work. Further, Mr. de Grave suggested that the motor should be put in a dry place, free from dust; they all knew that some collieries were dry, and how little dust was found in others. If the commutator was dust-proof, both commutator and brushes would be free from dust. The author suggested that belts and slow-speed engines should be employed instead of high-speed driving engines. He (Mr. Stokes) had seen both in use, and preferred the use of belts and slow-speed engines. Another suggestion was that the commutator should only be wiped with clean cotton-rag, just moistened with mineral oil. This was a very little matter, but a very important one for the commutator. He had been in an engine-house, and seen a man grease the

commutator with a bit of grease and dirt on his finger from the connecting-rod of the engine. He had been in other engine-houses, where careful men had been attending to the commutator, and everything had been clean and in excellent working order. There the attendant wiped his hands on clean pieces of cotton waste and put a little vaseline on the commutator, so that it was kept in excellent order. These details were seemingly unimportant, but were in fact most important. He was of opinion that vaseline was by far the best lubricant for the commutator. Mr. de Grave mentioned the burning-away of one of the commutator-bars, and put it down to jumping and vibration of the motor. This might be one cause, but there was a question whether the quality of the material was uniform throughout the bar. Many members had put down electric-plants, more were thinking of doing so, and the practical suggestions contained in this paper would prove very valuable, and when a motor or dynamo was out of order, the users would be able to find out the cause, and possibly remedy the defect.

Mr. M. DEACON said that the Davis and Stokes motor overcame a greater difficulty than that of dust, namely, the question of sparking in fiery mines. In this motor the commutator was inverted, the plates being on the inside of the commutator instead of the outside, and the end of the motor was closed by a cover having dust-tight joints, thus rendering it perfectly impossible for any gas to enter or to come out of the commutator if it were possible for it to get in.* One of these motors had been in operation for four years in a pit worked with safety-lamps, and it had worked extremely satisfactorily the whole time. It was used to drive a pump, where the load was pretty constant, but the head was not very great. The danger usually attendant on sparking was avoided by the commutator being enclosed, and it had been somewhat astonishing to him that this motor had not made greater headway. He had not heard of it being used for very high voltages, but he should like to see one running at 500 or 600 volts. He saw no reason why it should not work as well as the one referred to, which was a 120 volts machine.

Mr. H. R. HEWITT (Derby) said that there was a dynamo machine fitted with the Sayer commutator coils (which was non-sparking), although not enclosed in any way. The commutator was completely exposed, and ordinary brushes were used.

Mr. G. J. BINNS (Netherseal) said that Mr. de Grave's paper was valuable, especially as it contained perhaps the first admission from an

* *Trans. Fed. Inst.*, vol. iii., page 192.

electrician that an electric machine could go wrong. There had been a suspicion that something might go wrong, and sometimes even had gone wrong; but as a rule electricians would not admit that there was any likelihood of such an occurrence. This paper would show that electric machinery was not quite so near perfection as had been represented.

Mr. G. E. SMITH (Nottingham) asked Mr. de Grave what influence the field-magnet had upon sparking? The armature and commutator had both been spoken of in connexion with sparking, but might not the field-magnets in some cases cause sparking?

Mr. DE GRAVE, in reply to Mr. Smith's question, said that in order to explain the relation between field-magnet design and sparking, it would be necessary to go thoroughly into the question of the design and construction of dynamos. The design of the field-magnet played an important part in the prevention of sparking. In his paper, however, he referred only to machines of modern design, in which, so far as the management was concerned, field-magnet design did not come in.

The PRESIDENT agreed with Mr. Stokes that Mr. de Grave's paper was a very valuable one. The subject was treated in a practical way, and the paper would be useful to a much greater extent than previous communications on electricity. In his experience of electric-motors there had been some little sparking, of which he had taken note. A serious accident which recently occurred in London had been accounted for by the presence of sodium. Perhaps Mr. de Grave could explain how the sodium got there? He then moved a vote of thanks to Mr. de Grave for his interesting paper.

Mr. DE GRAVE said that sodium was formed by electro-chemical action on the insulator. Moisture covered the insulator, and electricity in leaking over formed the sodium from the salt in the glaze of the insulator. Then water in larger quantities fell upon this sodium—a drop was enough—and as an ordinary result, flame was produced.

Mr. A. H. STOKES, in seconding the vote of thanks, said that the discussion of this paper should be adjourned. There were points not mentioned that the author would perhaps consider before the discussion was resumed. Sparking was one of the points, especially if the motors were taken to drive holing-machines or to work in the mine. Sparking was to be dreaded from the very best machines, and there was the further danger of getting a spark from the wires when uncoupled, or by the cutting in two or fracture of the cable. It was dangerous to have a safety-lamp in the pit if a man put his pick through it, and it was equally dangerous to have electric wires if a man misapplied them.

Alternating current machines had been used in America for several years. There was, however, this objection to their use, that another machine was required to start them.

The vote of thanks was unanimously accorded.

Mr. EDWARD THOMPSON (Nottingham) exhibited and explained appliances for the better securing and setting steel girders when used for supporting the roof in mines. These comprised (*a*) an iron chair, for use with steel girders in main roads when supported by wooden props. The chair is made to fit upon the end of a steel girder, and on the underside of the chair a bed is formed to receive the top of the wooden prop. (*b*) An iron clip, intended for use with steel girders when supported by steel girder props. The clip is made in two pieces, which fit on either side of the girder-bar and prop, and on being bolted together, securely holds one to the other. (*c*) A wrought-iron holdfast, which is intended for the same purposes as the clip, but for use with heavier girders. The holdfast consists of a wrought-iron band, which fits on the outside of the steel bar and prop. A bolt is passed through the holdfast under the girder at the face of the head of the prop.

The meeting was then closed.

THE NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,
HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,
APRIL 27TH, 1895.

MR. JOHN DAGLISH, PAST-PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting and reported the proceedings of the Council at their meetings on March 30th and that day.

The proceedings of the Council of The Federated Institution of Mining Engineers were reported.

The following gentlemen were elected, having been previously nominated:—

MEMBERS—

Mr. GILBERT LITTLE, Manager, Engineering Works, 3 and 4, Lime Street Square, London.

Mr. WARDLE STEEL, Manager, Willington Colliery, County Durham.

Mr. W. DUNCAN WILLIAMSON, Chemist and Metallurgist, Mexican Gold and Silver Recovery Co., Ltd., 2A, Calle de la Providencia No. 7, Mexico, D.F.

ASSOCIATE MEMBERS—

Mr. CHARLES EDWARD HUNTER, Coal Owner, Selaby Park, Darlington.

Mr. A. LARSEN, Explosives Agent, The Carbonite Syndicate, Ltd., 24, Queen Victoria Street, London, E.C.

Mr. JOHN MARSHALL, 108, Poolstock Lane, Wigan.

ASSOCIATES—

Mr. JOHN NAISBITT, Overman, No. 48, Tudhoe Colliery, Spennymoor.

Mr. JOHN RAMSAY, Under Manager, Tursdale Colliery, Ferryhill.

STUDENTS—

Mr. HERBERT CHAPMAN, Mining Student, Hebburn Colliery, Newcastle-upon-Tyne.

Mr. NORMAN THORNTON, Mining Student, Pelton Fell, Chester-le-Street.

DISCUSSION UPON THE "REPORT OF THE PROCEEDINGS OF THE EXPLOSIVES COMMITTEE."*

The SECRETARY read the "Supplement containing the Results of Experiments with Westfalit, and a Further Series of Experiments with Detonators."

Mr. M. WALTON BROWN stated that the Council had considered the following question, which Mr. John Wilson, M.P., had asked in the House of Commons, and Mr. Asquith's answer, and had decided to offer to the Secretary of State for the Home Department the use of plant erected by the Institute for testing explosives to be used in mines :—

Mr. John Wilson (Durham) asked the Secretary of State for the Home Department whether he has decided to utilize the apparatus for testing Westfalit, erected at Messrs. Pearson & Knowles' collieries, near Wigan, for testing all other safety explosives used in mines, and whether he is aware that efficient apparatus has been erected by the North of England Institute of Mining and Mechanical Engineers, at a cost of about £500, and which has been seen and approved by several members of the Royal Commission on Explosions from Coal-dust, and, if so, whether he will take steps to secure that any experiments with explosives now used in mines shall be carried out by gentlemen not connected with the manufacture of explosives.

Mr. Asquith said he had not decided to cause this apparatus to be utilized, but if there should be need of it he would be very willing to accept the loan of suitable apparatus from any quarter. He was aware of the apparatus belonging to the North of England Institute of Mining and Mechanical Engineers; in fact some of the inspectors of coal-mines had been assisting at their experiments. Until the passing of a Bill dealing with the use of explosives in dangerous mines it would be premature to initiate experiments with different sorts of explosives, but he was fully alive to the necessity of impartiality in any experiments that might be made, and he might add that he hoped very shortly to introduce a Bill.

Mr. JOHN HOUSE wrote that from the fact that so many and careful experiments had been made in various mixtures of fire-damp and air, in addition to those with coal-gas and air, he regarded the experiments conducted by the Institute as unique, and as the most valuable of any which have been made in Great Britain, from every point of view, with the exception, perhaps, of the weights of explosive used for each charge, which he thought should have been more nearly those used in actual underground work, namely, from two to twelve times the weights adopted by the Explosives Committee. These weights seem, at first sight, to show a greater variation in the weights per shot of explosive

* *Trans. Fed. Inst.*, vol. viii., pages 227 and 593.

used than practice itself would authorize. To prove, however, that there was this great latitude, and that the maximum charge which he had mentioned was actually employed in blasting in coal-mines, he would submit a few figures to show the weights of explosive used per shot for varying circumstances in the mines under his charge. In a coal-seam 2 feet thick, and in which the coal is strong, the weight of roburite used in each shot varies from 2 to 3 ounces; in a coal-seam 3 feet 9 inches thick, but of only a moderately strong nature, charges of 3½ and 4 ounces are ordinarily employed; in another coal-seam of 4 feet average thickness, but of a strong character, charges of 4, 4½, or 5 ounces, and occasionally of 6 ounces are used, according as circumstances of "end," "face," or "fast" sides determine. In roofing-down stone canches, the charges vary from 4 and 6 ounces of roburite in ordinary shales to 10, 12, and 14 ounces in strong metals; and in rock-roofs 12 and 16 ounces are the regularly allotted charges. He was of opinion that if those officials and workmen, who are so tenaciously prejudiced against any and all substances proposed to be introduced as substitutes for blasting-powder, are to be educated to the knowledge of the dangerous character of their favourite blasting agent, and converted to the views of those who desire to select the safest substitute extant, it is of the greatest possible importance that experiments should be conducted with weights more nearly resembling those actually adopted in underground blasting operations. As there is some possibility of the various explosives being judged from tests made in the experimental apparatus for ascertaining their relative safeties when fired in gas and air; coal-dust and air; and gas, coal-dust, and air mixtures, he desired to draw attention to the absolute necessity of fixing upon a series of conditions which would embrace all the points which practical mining engineers should require from any explosive before selecting it as worthy of confidence, or if none are deserving of complete confidence, before saying which explosive forms the nearest approach to the ideal. Whatever discussion may arise thereon, he (Mr. House) hoped that the outcome might be the framing in a clear and decisive manner of a series of conditions for fulfilment by safety explosives, or those claiming to be such, that a colliery manager may, by their aid, experience the least possible difficulty in pronouncing a verdict upon the merits of such explosives when the results of the necessary experiments are placed before him. He (Mr. House) thought that the essential conditions were as follows:—(1) The explosive shall be incapable of being fired except by means of purposely constructed detonators. (2) It shall be of such a character that it cannot easily take fire, and cannot

burn in the hole, in case of partial detonation. (3) It shall not be liable to become frozen. (4) It shall not be liable to exude a dangerous liquid. (5) It shall be capable of being manufactured of uniform quality and composition. (6) When used in the regular way with stemming or tamping, it shall be incapable of firing explosive mixtures of fire-damp and air, or coal-dust and air, or both combined, even under conditions of a blown-out shot. (7) The products of combustion shall be completely gaseous. (8) The fumes, or products of combustion, shall be neither poisonous nor inflammable. (9) The element of safety in the explosive shall be self-contained. (10) The explosive shall be an efficient blasting agent for use in either coal or stone. The reasons for the particular conditions which he (Mr. House) had here laid down, dealing with them in the order in which they appear above, were as follows:—(1) Accidents, many of them trifling, but some of them serious and grave, and often attended with loss of life, have been occasioned by explosives being of an inflammable character, so as to be easily ignited by means of even a spark, or by concussion such as would be occasioned by letting the explosive fall, or by a blow or shock; or by the explosive being liable to spontaneous decomposition; or being dangerously sensitive to rises in temperature. He was of opinion that the safest explosive, so far as the circumstances here enumerated are concerned, is that which could only be fired by means of a purposely constructed detonator. (2) This condition, which demands that no explosive shall be liable to take fire and burn in the hole in case of partial detonation, is one which, to all practical minds, needs no comment. Tonite was at one time used to some extent at several collieries in Lancashire, but the experience of those who have tried it is that on many occasions, when partial detonation only has taken place, the part not completely detonated has merely been set on fire, and has fizzed in the hole, until the stemming has yielded, and part of the burning cartridge has been projected from the hole, rocket fashion, in a blazing condition. This constitutes a dangerous feature not for a moment to be tolerated. Of course, if it be agreed that the first condition is valid, then the second condition is also included; but if the first condition is not considered to be important, the second condition then becomes an urgent necessity. He believed that other explosives, namely, those which are liable to freeze, might in a semi-thawed condition be liable to this mischance, and he understood that such occurrences had taken place. (3) One danger from freezing had already been mentioned; another danger arose from the possibility of the ill-informed, or careless operator, endeavouring to force a detonator into a frozen or partially thawed cartridge. The greatest

danger, however, was incurred in thawing the frozen explosive. He need not refer to the proper method of thawing frozen cartridges, or to the rude and hazardous expedients which have been resorted to for this purpose, and which have often ended in disaster. (4) Nitro-glycerine compounds all exude nitro-glycerine to a greater or less extent. And every element of safety disappears with the exudation of nitro-glycerine from the cartridge. (5) The inconvenience, which at times has been occasioned by one consignment of an explosive differing in composition and in quality from a former consignment of the same explosive, renders it absolutely essential that a safety explosive shall be capable of being manufactured of uniform quality and composition. Some of the ingredients used in the manufacture of explosives offer greater obstacles to uniform manufacture than others, with the result that sometimes one supply of explosive may be, as regards blasting efficiency, all that can be desired; and another supply may obstinately refuse to detonate, and is compressed into a hard, solid substance in the hole. This defect only appears in the Sprengel explosives. Should any other class of explosive be incompletely detonated, the strong liability is for it to merely burn in the hole as already mentioned. (6) Many have been, and many are, the claimants for the position of a safe substitute for gunpowder; and up to the time of the issuing of the Explosives Committee's unbiassed *Report*, many have been the doubts, and many have been the fallacies held with respect to the merits of the rival explosives. (7) It is alleged that the solid products of the combustion of gunpowder, dynamite, etc., in a finely divided state, when projected at a high temperature into the atmosphere of a mine, may ignite dangerous mixtures of air and fire-damp or coal-dust. If this statement be true, it immediately becomes essential that "the products of combustion shall be completely gaseous." (8) The condition that "the fumes, or products of combustion, shall be neither poisonous nor inflammable," will commend itself to all. Carbon monoxide is an exceedingly injurious gas, 1 per cent. being fatal to life, and smaller percentages cause great injury to the health when frequently inhaled; consequently explosives, which on perfect detonation produce this gas, should not be commended for use. Profs. H. B. Dixon and Vivian B. Lewes state that, when this gas is present in a slight proportion in the atmosphere of a mine, coal-dust explosions may easily be produced by a blown-out shot. Explosives which form nitric fumes should also be avoided, as this gas is also capable of causing considerable injury to health. Hydrogen and fire-damp should not be found amongst the products of combustion of an explosive. Sprengel explosives do

not, on complete detonation, produce inflammable gases, and in this respect seem to possess a great advantage over other classes of explosives. If some independent chemist would make analyses of the products of combustion of the various explosives, he would confer a great boon upon colliery managers who are engaged in the task of discovering the properties of the various explosives now on the market. (9) It is essential that "the element of safety in the explosive shall be self-contained," that is, the explosive shall not rely upon extraneous assistance for lowering the initial temperature of its ignition, such as special stemming-material or water-cartridges as used with several explosives. These outside safety appliances may not be used, or may fail when used at some crucial moment. (10) It is also essential that "the explosive shall be an efficient blasting agent for use in either coal or stone." An explosive possessing the meritorious qualification of safety in explosive atmospheres is of no value if, when introduced for practical blasting, it does not furnish efficient results. The foregoing remarks have been made with the intention of showing other directions in which the Explosives Committee might with advantage make further experiments. The experiments made at Hebburn colliery, apart from strict impartiality (which is not their least recommendation), possess intrinsic value of the highest order, firstly, by reason of the efficiency of the apparatus employed; and, secondly, because the various explosives have been tested in true fire-damp and air mixtures. He (Mr. House) had briefly summarized the important results of this series of experiments as follows:—That all the explosives (ammonite, ardeer powder, bellite, carbonite, roburite, and securite) produce flame in a naked or unstemmed state, when no gas is present, and also in presence of inflammable mixtures of both coal-gas and air, and pit-gas and air. That this flame in the case of every unstemmed explosive is capable of igniting inflammable mixtures of coal-gas and air, but that in similar mixtures of pit-gas and air, the flames of unstemmed bellite, carbonite, and roburite have not caused ignition in twenty-five trials, whereas in the same pit-gas and air mixtures the flames of unstemmed ammonite (first shot), ardeer powder (fourth shot), and securite (first shot) have produced ignition. That when stemmed with 1 to 8 inches of damp puddled clay rolled into cylinders, all these explosives are capable of igniting inflammable mixtures of coal-gas and air, but that in no single instance did any of them ignite inflammable mixtures of pit-gas and air. These results have certainly taken mining engineers by surprise, and some of them seem difficult to account for. Previous to the issue of the *Report*, he (Mr. House) was of

opinion, and many others, held similar views, that all explosives when unstemmed would fire inflammable mixtures of coal-gas and air, and he certainly held the conviction that all explosives when unstemmed would also fire inflammable mixtures of pit-gas and air (although he had had no facilities for proof in the case of the pit-gas and air mixtures. He was surprised to find that the flames of unstemmed bellite, carbonite, and roburite had not ignited inflammable mixtures of pit-gas and air in twenty-five successive experiments. If this excellent record be maintained by future experiments, these three explosives must necessarily occupy the front rank. His next surprise was furnished by the experiments with stemmed explosives. From experiments that he had conducted in inflammable mixtures of coal-gas and air, he was of opinion that at any rate the Sprengel explosives were safe with about 4 inches of stemming for a 4 ounces charge, and that the addition of 1 inch of stemming for every extra 1 ounce of explosive would safeguard heavier charges, and he certainly expected that similar results would occur in inflammable atmospheres of air and fire-damp. The seventh conclusion of the *Report*, viz.: "The emission of flame from a blown-out shot of a detonated high explosive is not prevented by the quantity or length of stemming used,"* seemed to be justified by the experiments made by the committee. Now, although he did not desire the slightest relaxation of the General Rules (in the Mines Act) with respect to explosives, and although he would strongly deprecate the use of an absolutely perfect explosive (if ever there should be such a thing) in the presence of fire-damp in mines, yet he must maintain that stemming as used in actual practice must give a high degree of safety against the emission of flame, even should a blown-out shot occur. Though firing several hundred shots a week with a high explosive (roburite) in the mines under his charge, he had only had two cases of blown-out shots in five years. This fact proved the greatly lessened chances of blown-out shots with high explosives, and the results of the two cases mentioned tend to prove the value of stemming in actual blasting work, even should the shot be blown-out. The first blown-out shot (above referred to) was ignited by the fireman with the electric exploder in the usual way, no flame was seen, and no trouble, inconvenience, or mischief followed. The second blown-out shot (above referred to) happened in a different way. The collier, according to his own admission, which all the circumstances of the place after the event tended to corroborate, was engaged in unramming a missed shot in which the wires of the electric detonator had

* Page 41.

been short-circuited, and had not exploded. The charge was 6 ounces of roburite, and when the shot had been unrammed to within about 8 inches from the charge, he tried to pull the detonator out of the hole by force. The detonator was thus crushed by being drawn violently into the compact stemming, and the charge exploded. The stemming was blown out and struck the collier in the hands and breast, his eyes, fortunately, not being affected. The man stated that he saw no flame at all, and the hot products of combustion and stemming-material, although they turned his skin a red colour, and caused a burning sensation, had in no way scorched him. His skin was also pitted all over with pellets from the ramming or stemming. The man recovered in a short time from the effects of the shot upon his breast, the most serious injury being the shattering of the little finger of the hand nearest the explosive when it exploded. He was of opinion, with respect to the inefficiency of the stemming found in the committee's experiments, that it was due to the smoothness of the bore of the cannon which (having been obtained by the drilling-machine and being in steel) afforded no comparison with the rough sides of a shot-hole made in coal or stone, the roughness affording a certain degree of grip to the stemming, even should it be blown-out. The result of the resistance so afforded would be to allow time for the absorption of some portion of the initial heat by the combustion-products, by the stemming and the sides of the borehole, thus assisting the reduction of the heat generated to a point below that of the ignition of gas and air. In the smooth bore of the cannon the stemming is but loosely placed, for fear of damage to the cannon should the resistance be too severe, and the stemming is hurled out of the cannon with as little resistance as a projectile from an ordinary gun. These circumstances operate to the disadvantage of all the explosives tested under conditions of stemming. He was of opinion that if the experiments with stemmed shots could be made under circumstances more analogous to those of the mine, different results would be obtained from those recorded by the committee, and that several of the explosives under consideration would be found suitable for use in inflammable atmospheres of either coal-gas or fire-damp. Of course experiments with this object cannot be undertaken in the mine, but he suggested that to obtain almost identical conditions with those of the mine, experiments should be made in a short stone-drift or tunnel driven into the rocks of a hillside, quarry, or even a natural cave, if a suitable one could be found. This might entail some expense, but the Government might be induced to contribute to, or even defray the whole costs of such experiments.

considering the mighty issues at stake. Should experiments be conducted in such a stone-drift, or tunnel, or cave, the great advantage would be that in lieu of the cannon, the shots would be fired in holes in the actual stone measures. In this case stemming would be employed as it is in mining operations, and if an occasional shot did not become blown-out as arranged for, but forced down the rock instead, the only trouble that would be occasioned would be the mere drilling of a fresh shot-hole. With regard to the greater degree of safety which the committee's experiments have proved shot-firing in fire-damp and air to possess over similar shot-firing in coal-gas and air, he might say that he was prepared in the abstract for such a result, but he was not prepared to find that any explosive could safely be fired unstemmed in inflammable mixtures of fire-damp and air, any more than he was prepared to find that any unstemmed explosive could be fired in explosive atmospheres of coal-gas and air. He had read with great interest the discussion upon this particular matter by some of the members. His opinion was that the phenomenon is not to be explained by the small difference of 34 degs. Fahr. in the ignition-points of the respective gases when mixed with air, but by the longer time during which it was necessary to apply heat to secure the ignition of fire-damp and air, as compared with the time taken for the same heat to ignite coal-gas and air. The retardation of ignition was, he believed, due to the sluggish nature of the fire-damp, partly owing to the presence of other combinations than marsh-gas in the fire-damp, and partly attributable to the character of the marsh-gas. If this greater degree of safety was due to nothing more than such a difference of 34 degs. Fahr., he agreed with Mr. Henry Hall that it would be but a slender thread to depend upon for safety, and one which might in some cases disappear altogether, and everyone would strongly condemn any disposition to take advantage of such a circumstance in underground work. His (Mr. House's) remarks were intended chiefly to show:—(1) That there is a danger of an explosive being valued according to its behaviour in the experimental apparatus for testing its safety in inflammable atmospheres, without reference to other conditions which the exigencies of mining demand. (2) Although admitting the supreme importance of the abilities of an explosive to withstand these safety-tests in inflammable mixtures of gas, coal-dust, and air, yet it is decidedly advisable that a complete series of conditions be framed to cover all liabilities to accident from every possible cause, and also to cover the requirements of actual blasting; and that explosives be considered as to their competency to comply with the whole

of these conditions. (3) That stemming as applied in practical mine-blasting is not comparable with that employed in the experimental cannon, and that the stemming in the former case yields, under the conditions of blown-out shots, greater efficiency in assisting the lowering of the initial temperature of ignition of the high explosives, enumerated in the *Report*, below that necessary to ignite inflammable atmospheres. (4) The necessity of experiments to thoroughly test the efficiency of stemming, in cases of blown-out shots in inflammable mixtures, under conditions strictly comparable with those of mining. In conclusion, it is evident that the explosives experimented upon by the committee form most valuable substitutes for the old fashioned blasting-powder, and although no single one of them is absolutely safe under all conditions, yet they are all safer to use than gunpowder, in the presence of inflammable mixtures; and although some of them seem to possess higher merits than others, yet any one of them, as regards safety, is infinitely to be preferred to gunpowder, which has caused such calamities in the past.

Mr. HAROLD BONSER (Leeds) said that some objection might be made as to the tests made by the committee with regard to the detonators used; it would only have been fair in testing ardeer powder to have used treble detonators, as recommended by the manufacturers. He was of opinion that whatever quantity of ardeer powder was used, the length of the flame was the same, the flame being due to the detonator and not to the explosive. The relative weights of the explosives used in the experiments, as determined by the results of the experiment in leaden blocks, would not commend itself to practical men in actual working; thus $3\frac{3}{4}$ ounces of ardeer powder was certainly stronger than 1 ounce of bellite, one of the results ascertained by the committee. The experiments had been very carefully conducted, and he had no grounds for taking material exception to anything contained in the *Report*. The Royal Commission appointed to enquire into accidents in mines recommended in their final *Report** that high explosives (such as dynamite or gelatine-dynamite), when used in dry and dusty mines, should be surrounded by water, as in water-cartridges, or similar contrivances. During the last ten or twelve years, the water-cartridge placed round an explosive had never been known to fail, and he ventured to suggest that the committee should extend their experiments to the use of water-cartridges.

Mr. W. C. BLACKETT asked if the conclusions given in the supplementary *Report* were in substitution of the original conclusions published by the committee?

* Page 115.

Mr. J. L. HEDLEY (H.M. Inspector of Mines) said that the conclusions had been remodelled, so as to render them more clear and concise.

The CHAIRMAN said the *Report* stated that although there was only a difference of 34 degs. Fahr. between the temperature of ignition of marsh-gas and oxygen, and coal-gas and oxygen, not one of the seven explosives, when stemmed, had ignited an explosive mixture of marsh-gas and air, whilst, on the other hand, all the seven explosives had readily caused ignitions of explosive mixtures of coal-gas and air.

Mr. R. LAVERICK (Rainton) said that an incident very recently occurred at one of the pits under his management. A deputy charged a shot-hole with carbonite, and when putting in the second handful of stemming accidentally cut the detonator-wires. He then put in another charge of explosive in front of the first, in the hope that when fired it would explode both charges. After firing, he carefully examined the hole with the view of searching for the first charge, to satisfy himself that it had been exploded. He took all the loose coal, as he thought, from the hole, and concluded that both shots had been exploded. The next morning, however, a hewer who was working in the place struck the charge with his pick and it exploded. The stemming consisted of broken stone, and the length of stemming between the two charges of explosive would probably not be more than 2 inches.

Mr. H. BONSER said in a case of this description it did not necessarily follow, if a shot did not do its work, that it had not been ignited. Recently, two shot-holes were charged with roburite, and the chargeman, as he thought, fired them simultaneously. He detached the cable, removed the magneto-electric exploder, and placed the handle in his pocket. His assistant and another workman then went to examine the place, and to ascertain if the shots had done any work. They halted on the way, about 60 feet from where the shots had been fired; they were about to proceed on their way, when, to their amazement, one of the shots went off, and only two simple circumstances saved their lives—the halt mentioned, and the fact that it was a breaking-down shot.

Mr. WM. JAS. ORSMAN (Wigan) said that he was very pleased to be present at the discussion that day, and congratulated the committee on the extreme care and exactness with which the experiments had been carried out. With regard to the further paper that had been read, he could bear out the conclusions of the committee that the incomplete detonation of nitrate-of-ammonium explosives did not render them more capable of igniting inflammable gaseous mixtures. Various weights of

westfalit up to 12 ounces had been fired unstemmed from a strong steel cannon into explosive gaseous mixtures. With No. 6 detonators, containing 1 gramme of fulminate of mercury, incomplete combustion nearly always took place, and no ignition of gas ensued. With Nos. 7 and 8 detonators, containing 1½ and 2 grammes of fulminate, when complete detonation was observed, the explosive mixture was always ignited. In comparing the action of detonators, some distinction should be made with regard to the different makes, which vary considerably in composition. Nobel detonators, which are used principally for explosives containing nitro-glycerine, contain from 25 to 30 per cent. of chlorate of potash, mixed with the fulminate of mercury, whilst ordinary detonators contain only a trace. The large proportion of chlorate of potash contained in Nobel detonators, whilst acting usefully in the case of dynamite, carbonite, and other nitro-glycerine compounds, which are more perfectly decomposed by the action of heat as well as shock, renders them unsuitable for detonating Sprengel explosives, for which intense shock only is required. Consequently, with nitrate-of-ammonium explosives, detonators specially made for them should alone be used. He was of opinion that the use of other detonators was largely responsible for the cases of incomplete combustion which occasionally occurred. In trying the action of detonators themselves in explosive gaseous mixtures, we may expect to find different results with the different makes. A detonator containing chlorate of potash should develop more heat than one containing pure fulminate of mercury, and would give off gases containing heated solid particles of chloride of potassium, which would act like the smoke from gunpowder. The product from pure fulminate of mercury would, on the other hand, be entirely gaseous. Nobel detonators seem to have been exclusively used by the committee in their experiments, and it might be advisable to repeat a few of the experiments with other qualities. With regard to the statement made by Mr. Bonser concerning the hang-fire of a detonator used with a cartridge of roburite, it seemed to have been due to the priming-composition having been faulty and slowly burning away (instead of exploding so as to fire the fulminate instantaneously). A good deal of discussion had taken place with respect to the term "flames" used in the *Report*. There was no doubt that light had been noticed at various points of the apparatus at some distance from the mouth of the cannon. This might be due to (1) incandescent solid matter; (2) incandescent gaseous matter; or (3) reflection. (1) Incandescent solid particles could only be produced by firing those explosives which contain inorganic mineral matter such as gunpowder

dynamite, gelignite, tonite, etc. With nitrate-of-ammonium compounds, the products being entirely gaseous, no solid particles would be produced from the explosive itself. (2) Incandescent gaseous matter. It has been shown that heavy gases at a high temperature will become luminous, but one can hardly understand that the temperature would be high enough to cause the escaping gases to become luminous. Mr. Saville Shaw had dwelt on the fact that the explosion of gaseous mixtures depended not so much on the temperature as on the time during which the temperature had to be maintained, and he instanced the experiment of burning gun-cotton without igniting gunpowder. In this case the cotton burns so quickly that the sudden rise of temperature fails to ignite the powder. The converse of this experiment shows that gunpowder will always light gun-cotton. If flames were really produced at the distances mentioned, it must mean that burning was taking place all the time the flame lasted, and this would not be a very instantaneous process. In fact this was the first time we have on record the possibility of producing a flame 30 feet long that would not ignite an explosive gaseous mixture. One would imagine that the rush of burning gases along that distance alone would by friction cause ignition. Unfortunately, the results were derived from eye-observations only, and it was a pity that the suggestions in Part I. of the *Report* were not tried, namely, placing tufts of gun-cotton in the path of the flame. As gun-cotton ignited at a very low temperature (below 200 degs. Cent.), and could be ignited by almost instantaneous application of heat, this would have been a most severe test. At the present moment, in the absence of further proof on the above subject, one leant rather to the idea that the light seen was due to reflection. Considering the dusty state of the air which must be produced owing to the disturbances caused by explosion, and by the fan, a ray of light entering from one end ought to be clearly visible at some distance in the otherwise darkened chamber, and the halo of light produced by the detonation of Sprengel explosives might easily be reflected and mistaken for flame. With regard to the other conclusions of the committee (*e.g.*, No. 3) we find that compounds or mixtures containing nitro-glycerine do alter very considerably with age, owing to the exudation of nitro-glycerine, which render the explosive more dangerous, whilst nitrate-of-ammonium mixtures absorb water and so become inert. The statement in clause 7, that increased lengths of stemming did not ensure increased safety, was not borne out by the experience of other experimenters, either when firing from a cannon or in practical work in the mine. An increased amount of stemming meant greater resistance, and consequently a greater amount

of work is done in the coal or stone to be blasted, and the escaping gases must therefore issue into the air at a lower temperature. In a smooth-bore cannon, if wet puddled clay be used, the clay will be ejected in a solid mass, like a bullet, whereas if a stemming of a powdery nature be used, work is done and heat absorbed by breaking up the stemming into fine dust. The amendment of paragraph 8, to the effect that the nitrate-of-ammonium explosives were excluded from the statement that the explosive could be fired by percussion, bears out a paper* presented to the members at an earlier date. It was then shown that mixtures containing nitro-glycerine could be totally detonated by means of a 59 pounds weight dropping on a cartridge from a height of 6 inches and upwards. With the Sprengel explosives, even with a 30 feet drop, no detonation occurred: a small portion only of the explosive being decomposed, owing to the heat evolved by the impact of the iron surfaces. It had been alleged that for various reasons the experiments conducted by the committee were without value. Considering the very small knowledge on the subject, such experiments must be of the greatest possible value, as regards both the exact methods adopted and what may be expected to take place under certain conditions. Such experiments must suggest, and have already started, new lines of thought in students of the subject, and the various points will be taken up and worked out by many experimenters. No explosive ever has and probably none ever will be invented that will be perfectly flameless, but with the Sprengel explosives we have blasting-agents perfectly under control, and relatively safer when compared with gunpowder than a safety-lamp when compared with a naked flame. The fact that, with one at least of the Sprengel explosives, millions of shots have been fired in England during the past eight years without causing an explosion of gas or coal-dust in the mine, places them in the highest position as to safety.

Mr. J. L. HEDLEY said that Mr. Orsman had raised an important point as to detonators, and this matter had also been referred to by some of the other speakers, who had stated that with the small charge of explosive used the ignition of the gas on several occasions had not been caused by the detonation of the charge but by the detonator, and that the committee were using too small quantities of the explosives. The committee had been using small charges, but these charges were actually being used in mines in this district. He was not prepared to say that the ignition of the gas had not been caused by the detonator, nor was he

* "The Detonation of High Explosives by Percussion." By Mr. W. J. Orsman. *Trans. Fed. Inst.*, vol. iii., page 574.

prepared to say that the detonator had caused the ignition, but if it were admitted that the detonator had caused the ignition of the gas, they were opening up a question which had not been approached hitherto, namely, the use of detonators in mines. It was one of two things. Either the explosive or the detonator ignited the gas. If it were not the explosive, it became a serious matter for consideration as to what was to be done with regard to the detonators. The detonators used in the experiments were obtained in the open market, and supplied to the users of explosives in the different mines of the kingdom. They were now told that if a larger proportion of an explosive were used, the flame from the detonator would be quenched; and another speaker stated that when a still larger proportion of an explosive was used, the flame ignited the mixture. It was possible that a certain weight of explosive might have to be used to kill the flame of the detonator. If that were the case, all the shot-holes must be made of one size and the same weight of explosive used in every case, and this would be going beyond the grounds of possibility. Explosives, when used, had to be used in different weights, and unless explosives perfectly safe under all conditions could be obtained, they could only come to the conclusion that it was not safe to use explosives in mines. Mr. Orsman had told the members that some detonators produced flame and some did not, but was Mr. Orsman prepared to say what kind of detonator should be used, and what mixtures should be used in their manufacture? Mr. Orsman also stated that it was impossible to make a flameless explosive. If that were so, why should the ignition of gaseous mixtures be attributed to the detonator in preference to the explosive? If a detonator was fired in a darkened room, flame could always be seen, and that flame might or might not be able to ignite gas. The experiments with detonators made by the committee showed that in a large number of trials the gas was not ignited, while in a few cases it was ignited: in each case the same kind of detonators was used. Another question raised by Mr. Orsman was with regard to the extra amount of stemming. The committee stated that an extra amount of stemming—from 1 to 8 inches—made no difference so far as the safety of an explosive was concerned, this being the results of the experiments. The stemming used was, as he had previously explained, stiff puddled clay; the pellets were not dropped in, but were rammed in in the usual way.* The clay used was similar in all cases, and the shots were always stemmed under the same conditions; and to ensure this they were all stemmed by the same member of the committee. Mr. Orsman seemed to attach great importance to the friction between the sides of the hole

* *Report*, page 31.

and the stemming, and in the experiments there had always been this friction. The holes had been stemmed in the same way as was now done in a large number of mines—perhaps not coal-mines, but certainly metalliferous mines—in the country. Plastic clay alone was regularly used in a large number of metalliferous mines in this district, and with the very best results. With respect to the detonator hanging fire, he had had two cases under his notice in this district, where it was said the detonator hung fire, and that the shot had gone off afterwards, in one case killing a man and in the other injuring one. If this were so, and Mr. Orsman was of opinion that such a possibility might arise, it only taught them that they would have to be very careful with the detonator, and it pointed to a danger which they had hitherto evidently not suspected and to which they must attach more importance in future, but he did not think it had any bearing on the experiments now under discussion. Mr. Bonser had contended that there was no difference in the amount of flame given off from cartridges of different weights of the same explosive. That might be so in the open air, but it certainly was not the case where shots were fired in the bore of a cannon. The makers of safety explosives stated that they must be confined before they would afford safe detonation. Mr. Bonser had also referred to the use of water-cartridges, which acted efficiently, if they were properly used, but there was always the chance of this not being the case. One of the speakers stated that the flames supposed to have been observed could only be reflections or halos, otherwise the gas would have been ignited; but this matter was fully explained at the last meeting by Mr. Saville Shaw.* That they did get a flame without ignition of the gas was evidenced from the result of the experiments with detonators. If a flame of short duration could be produced by a detonator without igniting the gas, he thought it might be conceded that it was possible also to obtain an elongated flame from an explosive without igniting the mixture. This, however, was a question of opinion, and always would be. The Committee had to trust to their eyesight in regard to the observation of flame. They were told that when small charges of explosive were used they were using too powerful a detonator. He would like to ask the gentleman who made that statement whether the explosives which they had been testing required different detonators for the different weights of explosive used. If so, it was reasonable to infer that with a small charge, they must use a small detonator, and with a heavy charge they must use a very large detonator, and this again raised the question of safety. Of course, if they were to admit that the detonator was

* *Trans. Fed. Inst.*, vol. viii., page 606.

responsible for the ignition, then the larger the detonator the greater the danger, and the only final conclusion would be that the explosive which required the least powerful detonator must be the safest, if it were proved that they were equally safe in themselves. A member had remarked that the experiments were of no value, because the conditions under which they were made were not the same as they were in the mine, but the same speaker admitted that such conditions might exist, though it would not be proper to fire a shot in the mine if it were in that state.* They were all agreed on that point, and if they were all and always to carry out the regulations applying to mines they would not have as many accidents as they had. The explosives had not been tested with the view of showing that any one was better than another, or that any particular one was safe; they only wanted to find out whether, under conditions that might exist in a mine, these explosives were safe or not. The same speaker demurred to a comparison between gunpowder and high explosives with the Trauzl method. The committee did not intend to compare gunpowder with high explosives, and the Trauzl blocks were simply used so as to make a comparison between the different high explosives.

The further discussion was adjourned.

DISCUSSION UPON PROF. STROUD'S PAPER ON "MAGNETIC DECLINATION AND ITS VARIATION,"† AND MR. J. HENDERSON'S PAPER ON "MAGNETIC DECLINATION IN MINES."‡

Mr. A. BEANLANDS (Durham) wrote that in the course of the discussion a statement had been made, with reference to the Ordnance Survey plans, which required correction. It was assumed that the four sides of each map or sheet corresponded with the true north-and-south and east-and-west directions. This was not precisely true—the north-and-south line passing through the principal station on Brandon Hill, with its continuation north and south, was the only marginal line in this district which coincided with the true meridian. The other north-and-south lines to the east and west of this were only approximate. In the case of the maps drawn to the scale of 6 inches to the mile, each sheet is laid down to contain a rectangular space 6 miles in length from east to west, and 4 miles in breadth from north to south. It was clear that if the line on the east of Durham, *i.e.*, 6 miles east of Brandon Hill, coincided with the true meridian, it must converge

* *Trans. Fed. Inst.*, vol. viii., page 593.

† *Ibid.*, vol. vii., page 268; and vol. ix., page 26. ‡ *Ibid.*, vol. viii., page 273.

towards the north pole, and the upper east-and-west line, must be shorter than the lower east-and-west marginal line, but as the upper, as well as the lower, margin was 6 miles in length, the eastern marginal line must be drawn not truly north and south, but diverging towards the east. Without pretending to extreme accuracy, he made the deviation of this line to be 7½ minutes, or ⅛ degree; and 15 minutes, or ¼ degree, for the marginal line farther east, *i.e.*, 12 miles from Brandon Hill. In like manner, if we take the Ordnance Survey sheets to the west of Brandon Hill, the marginal lines, 6 miles and 12 miles from the principal station, diverge ⅛ and ¼ degree to the west of the true meridian. Owing to the extreme difficulty of obtaining a loose needle bearing in many cases, as stated by Mr. Henderson, it seemed all the more important, wherever practicable, to obtain a direct connexion between the surface and underground surveys, such as was effected by means of the transit-instrument. He was not acquainted with the kind of shaft commonly met with in Cornwall, but should like to know whether, in case it was impossible to obtain a sight from the surface to the point desired, it would not be possible to set up a theodolite at the point where the inclination of the shaft changed.

Mr. JAMES HENDERSON (Truro) said that some remarks had been made referring chiefly to the almost absolute impossibility of using the compass-needle with accuracy in underground operations; he contested this statement because he was satisfied that in most places the needle could be used perfectly, and when applying the compass in a proper way he considered it a most admirable instrument. One gentleman said that he would not think of using a miner's compass for defining boundaries, but it was the only instrument which they did use in Cornwall. In the North of England they could take a line down the shaft from the surface and connect that with the underground survey, but in Cornwall the shafts were so tortuous that it would be impossible to take any line down them from the surface, and therefore they were compelled to trust to the magnetic bearings for intricate work. In mines where inches of ground were important, where lodes were perpendicular, they could imagine what efforts were made by surveyors to attain accuracy, but for the groundwork of all their surveys, whatever instruments they used underground, the best was the miner's compass. The theodolite was a valuable instrument no doubt, and the transit was a still finer instrument, but a transit, or even a fine theodolite, would be quite out of place in the Cornish mines with their small levels only a few feet high. How could they use a fine instrument, with mud and water falling upon it? In Cornwall they were obliged therefore to revert to the miner's compass; he would by no means recommend the use of two

instruments—one of first-rate quality at the surface and an inferior one underground. The same instrument should be used in both cases, even though it were of a somewhat inferior kind. Within the last year or two he had introduced what he considered to be a very valuable instrument for the purpose of traversing underground: connecting one's bearings off the traverse by means of a compass-bearing and carrying it on, on the theodolite principle through the mine, and the results of his experiments with the traverser wherever it had been used were most satisfactory, the holing of the shafts being most accurate. He had already had the pleasure of describing the instrument in the *Transactions*.* Mr. Henderson then exhibited and described the instrument referred to.

COAL MINES REGULATION BILL.

The SECRETARY (Mr. M. Walton Brown) read a paper on the "Coal Mines Regulation Bill."

The CHAIRMAN said the subject was one of considerable importance. The propriety of the Institute taking up such a subject had been considered by the Council that day, and it was understood that after the paper had been read the Council would further consider whether it should be published in the *Transactions*.

Mr. W. C. BLACKETT said it was important that they should have a knowledge of all technical matters which were brought before the Houses of Parliament. The reading of this paper was an innovation, and if it were admitted that this paper came within the province of the Institute, it must be understood that they would have to take cognizance of similar technical measures in future, or otherwise their silence about other bills might be misinterpreted.

Mr. T. E. FORSTER considered that the matter was not one coming within the province of the Institute.

Mr. J. A. RAMSAY considered that the Bill was one affecting mining engineers, and as such should be taken cognizance of by the Institute, inasmuch as recent legislation had practically placed the managers of collieries directly under the control of the inspectors of mines. He thought that any act of parliament which in any way affected the management of mines was a very proper subject of consideration by the members of this Institute.

Mr. J. G. WEEKS said the matter was one for discussion by the United Coal Trade and not by the Mining Institute.

Mr. T. E. FORSTER moved, and Mr. J. A. RAMSAY seconded a motion that a vote of thanks be accorded to Mr. M. Walton Brown for his paper.

* *Trans. Fed. Inst.*, vol. v., page 199.

DISCUSSION ON MR. FRANK COULSON'S PAPER ON "SINKING WITH ROCK-DRILLS."*

Mr. H. BONSER (Leeds) wrote that the use of high explosives had been so admirably described by Mr. F. Coulson in his paper, that no further reference is necessary, and all the members could refer to that paper in case any information was desired as to the sinking of shafts by means of high explosives. The economy of adopting the system explained in that paper cannot be too highly emphasized. In the subsequent discussion one of the speakers alluded to the fact that in the Durham district most of the pits required were sunk, and no doubt that was so, but it was equally true that in nearly every colliery it was found necessary to drive stone-heads or drifts or tunnels, and, no doubt, if the members would take Mr. Coulson's paper, and, instead of applying it to a shaft, would apply the self-same method to the tunnel or drift, equally satisfactory results would be obtained. For instance, in a drift progressing at the rate of, perhaps, 30 feet per week, if the holes were lengthened from 4 to 6 feet, and the unkeying-holes (corresponding to the sumping-shots of the shaft) were fired simultaneously, and the side holes were lengthened in the same way, the rate of progress with the same number of men employed would be increased to 45 feet in the same time. The saving would be effected by giving the explosive its full burden of work, and by the time saved in setting and changing drills, delay in firing, etc. The advantage of electric detonating-fuzes over ordinary time-fuzes was not only in igniting the shots simultaneously and the holes or charges, thus assisting each other, but also in the total absence of fumes and smoke from the burning fuzes, which were greatly in excess of the fumes or smoke created by the explosion. Too much emphasis cannot be placed on this point, as the saving in cost of driving a stone-head or tunnel may be from 30 to 50 per cent. of the total. There was just one word of warning to sinkers in wet ground which may require tubbing, that the side or cauch holes, if fired simultaneously, may have so disturbing an effect on the strata as to nullify the effect of the tubbing in keeping the water back. In such cases it is better to fire the shots in succession.

The meeting was then closed.

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* *Trans. Fed. Inst.*, vol. viii., page 17.

NORTH STAFFORDSHIRE INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD AT THE NORTH STAFFORD HOTEL, STOKE-UPON-TRENT,

APRIL 8TH, 1895.

MR. E. B. WAIN, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and confirmed.

The following gentleman, having been previously nominated, was elected:—

ASSOCIATE—

Mr. JOHN LLOYD KNIGHT, Under Manager, Adderley Green Collieries, Stoke-upon-Trent.

Mr. JOHN HEATH read the following paper on "A Modified Form of Longwall Working as Applied to Thin Seams of Moderate Inclination":—

A MODIFIED FORM OF LONGWALL WORKING AS APPLIED TO THIN SEAMS OF MODERATE INCLINATION.

By JOHN HEATH.

The system of working which the writer is about to briefly describe, though perhaps not new, has, so far, not been previously applied in this district.

In a district like North Staffordshire, the economical working of thin seams must always be one of the difficulties that confront a colliery manager, and a description of any method that tended to reduce the cost of getting coal would be valuable.

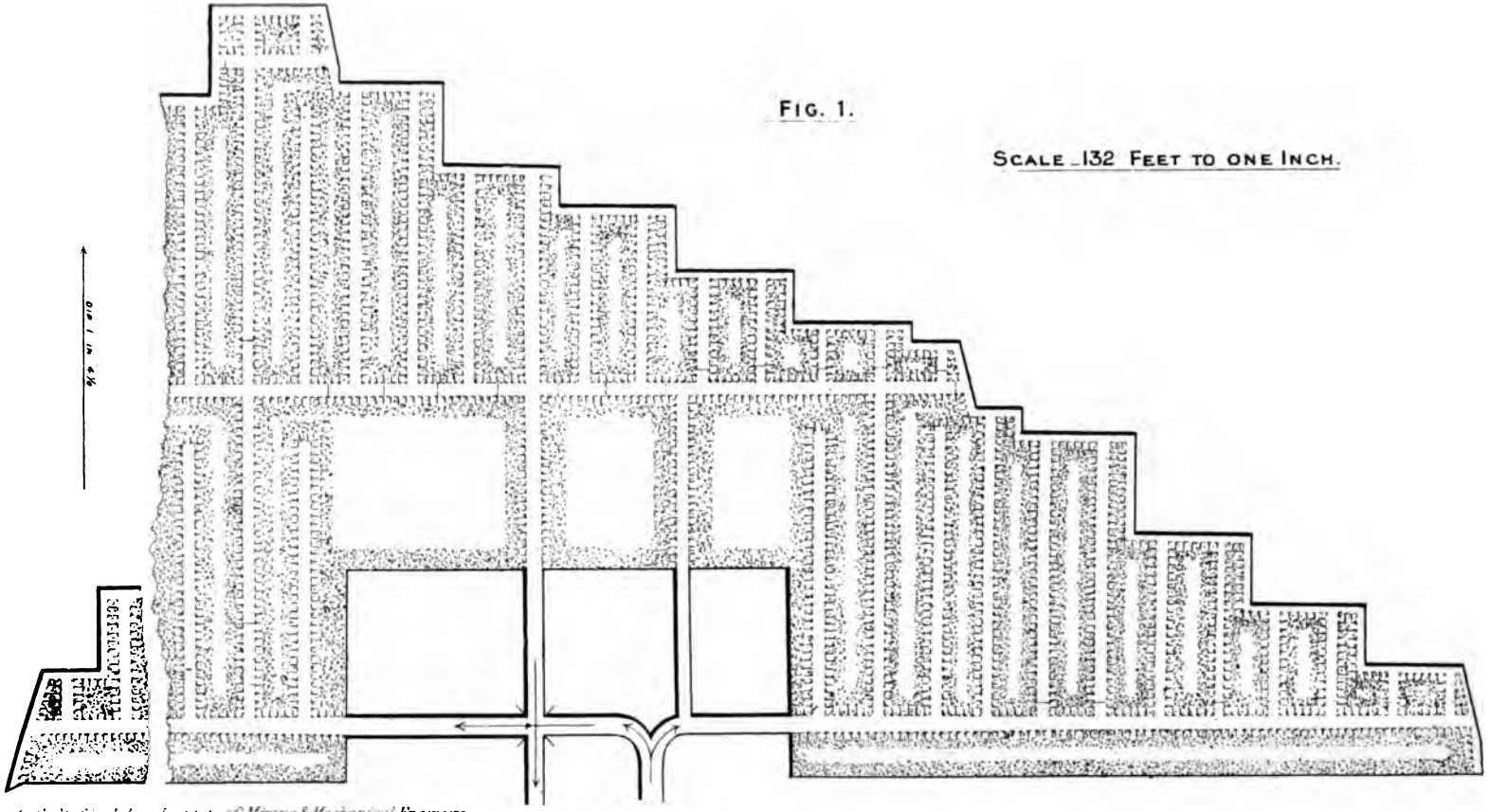
In a thin seam, the dead-work was always necessarily costly, and it was by seeking to reduce the amount of ripping that attention was directed to what the writer may call the "spunney" system.

As an example of its practical feasibility he may cite the two feet coal-seam at Sneyd colliery. In this seam, with an average thickness of 2 feet 1 inch, there was room to pack all the dirt from roads 6 by 10 feet used for levels, 5 by 7 feet used for spunneys, and from the main jigs where the ripping was 3½ feet thick; the hard ripping dirt was used for building the walls at the sides of the roads, no stone whatever being sent to bank. Where the coal-seam had thinned down to 12 inches, they were able to work it, although, of course, in an extreme case like this, the working cost almost prohibited competition against coal produced from seams varying from 3 to 8 feet in thickness.

The main object of the system is to enable the coal to be loaded into the waggons without taking them along the face, and to accomplish this object the jigs or spunneys, as they are termed, are placed at a distance of only 42 feet apart from centre to centre. The loader reaches the coal as far as possible, and in the remaining 5 or 10 feet the coal is turned back by the coal-getter, as is the usual custom in other methods of working. The waggon is not unhooked from the chain, the front wheels being simply dropped over the end of the rails and the two brake-wheels screwed up. In the case of the two feet coal-seam, it is thus possible to load a waggon standing 33 inches high on the rails, and holding 10 to 12 cwts. of coal at the face of a seam measuring only from 23 to 25 inches in thickness.

FIG. 1.

SCALE 132 FEET TO ONE INCH.



The spunneys are arranged in pairs, with a single line of rails in each, the empty waggon travelling first in one and then in the other of the pair, the chain passing along the face between the road-ends. They are driven in this form for a distance of 240 feet, then a top-level is formed, and one of the spunneys is used as a standing jig. These jigs are laid with a single road at the top and bottom, so that the spunneys need not be made of extra width except at the pass-bye. One stand jig will, as a general rule, serve for five or six pairs of spunneys, the tubs being jigged in runs of two or three, as the several lengths of stand jigs are connected up (Fig. 1, Plate XIII.).

In breasting out the levels a solid pack of at least 36 feet wide is placed on the deep-side, and one of 30 feet wide on the head-side. Pack-walls are built at right angles to the level, at intervals of 5 feet, and the loose dirt between them is rammed solid up to the roof. When the pinnings are built diagonally, as is often the case where the deep side of the breasting is pushed in advance of the head-side, and the pack-walls form an acute angle with the level, they are found to push out much more readily than when taken square up. Chocks are built in at the head-side at the corners of each spunney, and on the deep-side midway between the two spunneys, so as to come alternately on either side of the road.

More stress has been laid upon the question of packing than will perhaps be thought necessary, but proper packing, in the first instance, is of the utmost importance, and, in the writer's opinion, cannot be emphasized too strongly, for not only is it essential in keeping a good level, but it prevents air from leaking through into the wastes, with the accompanying evil of such leakage—gob-fires.

In addition to diagrams being shown, a model of the method of jigging employed was exhibited in the room.

The PRESIDENT thought that this plan, showing a method of working a coal so thin as 2 feet, was particularly interesting. The time had not yet come when they had to work a very large quantity of coal so thin as that, but they did not know how soon they might have to do so, and it would be well to consider the best method of doing it. The system adopted at Sneyd colliery appeared to be somewhat similar to that adopted in the best collieries in the Midland district, where it was the practice to make gate-roads, and cut them off with cross-

roads at such intervals as were necessary in accordance with the nature of the roof. Mr. Heath had not indicated the nature of the roof of the coal-seam which was being worked by this system, but it struck him (the President) that if, in a system like that, they had a hard roof to contend with, they would be put to serious expense in yard work and ripping. That was an idea perhaps based on imperfect knowledge of the system as worked at Sneyd colliery. He had an opportunity two or three years ago of seeing the system at work, and it seemed to him a practicable method of getting thin seams. It appeared, however, rather a misnomer to call this kind of working "longwall." His idea of longwall was a continuous face of work, and in this case, where there was only a total length of continuous face of 84 feet, it could scarcely be called longwall.

Mr. HEATH said that the roof in the seam which he had been working on the principle described was good hard metal.

Mr. BAILES enquired whether there was a pair of spunneys, with a single road in each case?

Mr. HEATH—Yes.

Mr. BAILES said that it was like a jig with double road and packing between.

Mr. W. BRIDGETT said that there appeared to be too many roads to make and maintain for the small amount of coal got. A system he remembered seeing in use for working thin seams of both coal and ironstone was to make one jig road for 250 or 300 feet of coal-face; and the coal was danned out to the jig and reloaded into waggons, and probably by taking up a little floor in the drifts sufficient height could be maintained to allow of this system being adopted.

Mr. PREST enquired why the face was stepped in preference to a straight face, and was it necessary to commence coal-getting at once? He presumed that the whole of one drift or gate operated by a pair of spunneys would be in the hands of one contractor, otherwise it might occur that the men in one spunney would not be able to get out their coal when the men in the other spunney were idle. There was an undoubted advantage in breasting-out the levels in thin seams, and getting out as much of the coal to the deep side of the level as possible, if the time element did not enter into consideration. Whether the system described by Mr. Heath as a whole was more profitable than the customary plan of working would depend on a variety of circumstances that could only be settled by actual experiment. There would be a saving of cost in not being compelled to rip the top along the face;

against this, however, must be set the increased number of spunneys or jigs required, and the cost of converting some of them into main-jigs when the top-level had been formed, as well as the probable decrease in the percentage of round coal, owing to the man being compelled to turn the coal over several times before being in a position to load it into tubs.

Mr. B. WOODWORTH said that some time ago a paper was read at a meeting of mining students respecting the working of thin coal in the North of England on similar lines to those adopted at Sneyd colliery; but in that case, being practically on the level, the faces were continuous. Mr. Heath's diagram reminded him of the truth of a remark made some twenty years ago by a shrewd colliery proprietor in this district when talking about coal-cutting machinery for such seams, namely, that the best coal-cutting machines were good colliers at reasonable wages. The present paper seemed to prove it when arrangements were made to suit the occasion.

Mr. GREGORY said that was a system of panel work.

The PRESIDENT believed that this system had been worked at great depth.

Mr. HEATH said that the workings were at a depth of 1,950 feet.

The PRESIDENT asked whether the coal was got practically without holing?

Mr. HEATH answered in the affirmative.

The PRESIDENT asked whether Mr. Heath had any idea what would be the effect of working this system in a narrow working where, unless he had a sufficient length of face open to allow of the roof breaking the coal, he would be obliged to blast?

Mr. HEATH replied that the coal of one drift was blasted.

The PRESIDENT said that the system seemed to him a distinct improvement in working a seam with tender roof, as they would not have much trouble through the working being constantly under new roof. The fact of there being in this system a cutting or carving would not seriously affect the working cost, as the carving, although 45 feet long, was all worked off in a month. He saw some six years ago how well these carvings kept. They were able to get over the ground quickly, and the timber cost was practically *nil;* but he still thought that the dead-work cost was heavy in proportion to the total working cost.

Mr. FROST said that the spunneys were 42 feet apart. Up every main-jig there were six sets of spunneys. There was no timber in the jigs.

Mr. PREST moved that a vote of thanks be accorded to Mr. Heath. The motion was agreed to, and the discussion was adjourned.

Mr. J. J. PREST read the following paper on "Modern Coal-tipplers:"—

MODERN COAL-TIPPLERS.

By J. J. PREST.

The many different designs of coal-tipplers are evidence of the great advances made during the past ten or fifteen years in the efficiency and improvement of colliery-plant, and particularly in the appliances adopted for screening and cleaning coal. Formerly a long array of fixed-bar screens, each fitted with its own tippler, was seen at all coal-drawing shafts, but now the entire output of a large colliery is concentrated at one point, and passed through a single power-driven tippler at considerably less cost and with greater efficiency than was the case under the old order of things.

The following particulars are given of coal-tipplers that have come under the observation of the writer:—

1.—*The Heath and Woodworth End Tippler* (Figs. 1, 2, 3, and 4, Plate XIV.).—This tippler consists of a pair of parallel cast-iron wheels, A^1 and A^2, having bearings or journals at their centres, and mounted on cast-iron pedestals, B, at either side. The wheels or side-frames are attached together by angle-iron distance-pieces, on to which is secured a wrought-iron plate, C, carrying the rails for the tub, and the axle-catches or blocks, D^1, D^2, as well as two vertical plates, R^1 and R^2, having pieces of angle-iron riveted on the upper sides. The axle-catches or blocks, and the pieces of angle-iron, are used for the purpose of keeping the tub upon the tippler whilst being emptied of its contents. The wheel, L, runs loose on the same shaft as the wheel, A^1, and is always in motion, being driven by a chain or rope from a countershaft connected with the screen-engine. On the other wheel, A^2, is fixed a bell-crank lever, M, coupled by a rod to small arm, and a weigh-bar, N, and arm, N^1, for the purpose of raising or lowering the axle-catches, D^2.

The axles of the tub, on being placed on the tippler, depress the first pair of catches, D^1, then gear with one of the four teeth of a star-wheel, *a*, giving it a quarter turn, and so by means of thumb-shaped projections, a^1, which gear alternately with the gib, *b*, formed on the slide, P, move the latter and the small arm, N^1, and weigh-bar, N, backwards; thus allowing the weighted ends of the tub axle-catches, D^2, to fall down, and

the upper ends to come into contact with the axle of the leading pair of wheels. The axle of the last pair of wheels is secured by the first pair of catches, D^1, being forced back to their normal position by means of the spring, E, after the tub has passed over them.

The tippler is put in motion by pulling over the lever, F. This operation draws back the stop, G; the spring, H, acting on the bell-crank lever, J, forces the slide, K, into gear with the teeth of the wheel, L, and causes the tippler to rotate in the direction shown by the arrow. The lever, F, is brought back into position for stopping the tippler by means of the spring, H^2.

In the process of making one revolution, the bell-crank lever, M, is brought into contact with the wiper, O, and through the medium of the arm, N^1, and weigh-bar, N, the slide, P, is pushed under the axle-catches, D^2, so that the upper ends are depressed below the level of the axles of the leading pair of wheels, allowing the tub to pass off the tippler, when the latter is brought to a stand. As soon as the bell-crank lever, M, is brought into contact with the wiper, O, the bell-crank-lever, J, also comes into contact with the stop, G, drawing the slide, K, out of gear with the driving wheel, L, and so stops the tippler.

This arrangement of tippler has been adopted at the Sneyd colliery, Burslem, Staffordshire.

2.—*The Wood and Burnett Rotary Tippler* (Figs. 5, 6, and 7, Plate XIV.).—This tippler is designed to be driven by a chain and sprocket-wheel, or by toothed gearing; the figures illustrate the former arrangement, but the starting and stopping-gear are the same in either case. The tippler consists of a pair of circular cast-iron wheels, A, riveted together by angles and bars in the usual manner, and supported on four cast-iron rollers, B. A countershaft, C, driven by the screen-engine carries the sprocket-wheel, D, attached to a friction-clutch, E, by means of which the tippler may be put into gear with the shaft, C. The friction-clutch also forms a brake-wheel, a wrought-iron strap passing round it for that purpose. The sleeve of the friction-clutch and the brake-lever are moved by means of a weighted bell-crank lever, F, so arranged that when the end, *a*, is depressed the brake is applied and the friction-clutch taken out of gear; and when released, the weight causes the brake to be released and the friction-clutch to move into gear.

The ends of the levers, H and F, are in the same vertical plane, and connected together by means of a bolt, K, so designed that it can be lengthened or shortened by the wedge, L, being thrust into or withdrawn

from a box formed in the centre of the bolt. The movement of the wedge, L, is controlled by means of a hand-lever, M, counterbalanced by weights in such a manner that it assumes the position shown in the figures. When the wedge, L, is to be pushed into the box, the end of the lever, F, is depressed, putting on the brake and throwing the friction-clutch out of gear, and, by the same movement, stopping the tippler.

The tippler is started by means of the hand-lever, M, so that the wedge, L, is withdrawn from the bolt, K; the latter is thus shortened, allowing the weight on the lever, F, to fall, by which the brake is released and the friction-clutch put into gear. At the same time the lever, G, falls clear of the stop, I, and the tippler at once commences to rotate. The hand-lever, M, is then allowed to fall back and to push the wedge, L, through the aperture in the box, thus lifting the end of the lever, G, into position ready to engage with the stop, I, when the revolution of the tippler has been nearly completed. The lever, G, is pushed down by the stop, I, and so applies the brake, and takes the friction-clutch out of gear automatically and simultaneously. An auxiliary attachment, consisting of a treadle, lever, and rods, *c*, *d*, *e*, is applied, so that the tippler can be stopped at any part of its revolution if necessary. A plate, P, is attached to the tippler with the object of reducing the breakage of coal whilst it is being emptied on to the screen.

This arrangement of tipplers have been adopted by the South Hetton Coal Company, Limited, Sunderland.

3.—*The Wood and Burnett Simplex Rotary Tippler* (Figs. 8 and 9, Plate XIV.).—This tippler consists of the usual form of cylindrical frame, A, supported by two pairs of rollers, B and C. The rollers, B, are keyed on to a shaft having a continuous rotatary motion, whilst the other rollers, C, can turn freely with the tippler, and do not act in a driving capacity, and as long as the frame, A, rests upon the rollers, B, friction will cause it to rotate with them. The principle of this tippler is, that, when at rest, one side of the frame, A, is lifted clear of the rotating rollers, B, and when rotation becomes necessary the frame is lowered down upon these rollers.

For this purpose two different designs are shown. Fig. 8 shows a brake-block, D, operating on the rim of the tippler-frame, A, by means of a rod from the lever, F, through a toggle-joint, E, this lever being kept in position while the tippler is at rest, by the stud, G, on the frame, A. The weigh-bar on which the lever, F, is secured, is long enough to reach to the other end of the tippler, and another brake-block is similarly applied on the other rim of the wheel or framework of the tippler. The

tippler is started by pulling over the lever, F, and disengaging the stop-pin, G. The tippler being thus lowered on to the rotating rollers, B, at once commences to revolve, until the stop, G, again comes in contact with the starting lever, F, when the apparatus is stopped.

In the modification shown in Fig. 9, the brake-blocks, D (attached to the ends of the tippler-frame, A), mount a slightly inclined plane, H, the lower end of which is hinged to the framework carrying the tippler, the higher end resting on an eccentric piece, or cam, K, connected with the starting lever, F.

The tippler is started by placing the lever in the position indicated by the dotted lines, the inclined planes, H, fall, and the tippler-frame is lowered on to the friction-wheels, B, and rotation commences.

A feature of the simplex tippler is that the centres of gravity of the tub and of the tippler-cage do not coincide, the tub being set a little on the teeming side so as to facilitate the starting of the tippler.

4.—*The Lührig (or Blackett) Side-tippler* (Figs. 10, 11, and 12, Plate XIV.).—This tippler has the usual type of cylindrical frame, and is supported on four cast-iron friction-rollers. The rollers, A A, are keyed upon the shaft, B, which also carries a belt pulley, P, driven by the screen-engine or other motor. This pulley runs loose on the shaft, and forms one-half of a conical friction-clutch (Fig. 11). The starting-and-stopping gear is chiefly connected to a weigh-bar, C, placed at one side of the tippler. Upon this shaft are keyed three levers; a hand-lever, D, placed in a convenient position for the banksmen; a counterbalance lever and weight, E; and a lever, F, placed opposite the centre of the tippler. The hand-lever, D, is continued downward beyond the shaft, C, and its lower extremity is connected by an adjustable link, *a*, and a bell-crank, *b*, to the sleeve of the friction-clutch. The gibbed lever, G, works loose on the shaft, B, and any lateral movement is prevented by the two collars, *c c*, this lever being placed opposite the lever, F, to which it is connected by a rod, *d*. The gib upon the end of the lever, G, engages with a projecting plate, *e*, fixed to a cross-bar on the tippler-frame; there is also, attached to the lever near the gib, a hinged sneck, *f*, which is intended to prevent the tippler from rebounding when brought to rest by the gib. The end of the lever, F, works in a slot formed in an arm, H, which is hinged to a post fixed at the side of the tippler. A pawl, *g*, is hinged to the arm, H, and is arranged to engage with a stud, *h*, on the lever, F. A projecting plate, *i*, is fixed to a cross-bar on the tippler-frame, so that it lifts the end of the arm, H, as the tippler turns round.

On a tub being placed in the tippler, the hand-lever, D, is pushed over, until the pawl, *g*, engages with the stud, *h*, on the lever, F, thus keeping the gib-ended lever clear of the tippler-frame, and putting the friction-clutch into gear. When the tippler has nearly completed a revolution, the arm, H, is lifted by the plate, *i*, liberating the lever, F, and allowing the counterweight to fall and move the levers into such a position that the friction-clutch is out of gear, and the gib-ended lever, G, engages with the plate, *i*, on the tippler-cage, bringing the tippler to rest in the correct position for the next tub to be run into it. The arm, H, is supported by a chain, which is continued from the bellcrank, *k*, in a suitable position, so that the banksman may, by pulling it, throw the driving mechanism of the tippler out of gear, and thus, if necessary, stop the tippler at any point in a revolution.

This arrangement of tippler has been adopted by the Charlaw and Sacriston Colliery Company, Limited, Durham.

5.—*The Silvester Power-driven Tippler* (Figs. 13 and 14, Plate XV.).—This tippler consists of a pair of cylindrical cast-iron rings, A, A, resting on four cast-iron rollers, B, B, supported on stud pins, C, C, well fitted into bored holes in the side girders, D, D. The whole arrangement is carried on a cast-iron girder frame, D, D. The cast-iron rollers, B, B, run in a groove made in the rims of the rings, A, A. Two similar rollers, E, E, are keyed upon the opposite ends of the shaft, F, resting on sliding carriages, to which motion is conveyed by means of a belt or rope, driven by a constant-running countershaft. When a tub is placed on the tippler, and the lever, H, pressed down, the friction-rollers, E, E, are brought into contact with the rims, A, and the tippler revolves so long as the contact is maintained.

This arrangement of tippler has been adopted by the Chatterly-Whitfield Colliery Company, Limited, Norton-in-the-Moor, Staffordshire.

6.—*The Shelton Tippler* (Figs. 15, 16, and 17, Plate XV.).—The tippler consists of a framework or cage made of two steel angle rings connected by cross bars, and supporting a length of tram-road, on which the tubs to be emptied are run. The rings, A, A, are supported by four cast-iron flanged rollers, B, B and C, C. The rollers, C, C, are keyed on the shaft, D, on which are also mounted the fast and loose pulleys, F and F^1. The strap connected with these pulleys is driven from a countershaft on the screen-engine or other convenient motor, and the size of the pulleys is arranged so as to give a speed of rotation to the tippler of about twelve revolutions per minute.

The starting-and-stopping mechanism of the tippler is connected to the shaft, G, fixed at one side of the tippler and at right angles to the shafts supporting the tippler-cage. On the shaft, G, are keyed three levers: a hand-lever, H, the end of which is placed in a convenient position for the banksman; a counter-balance lever, I, on which is fixed a weight; and the lever, K. The lever, K, has a gibbed-end, which engages (when the tippler is at rest) with the projecting plate, *a*, fixed to one of the cross-bars on the tippler-cage. Near to the gibbed-end is the small sneck, *b*, controlled by a spring, *c*, used to prevent the tippler from rebounding on being brought to rest by the gib. A pawl, L, is hinged to the framework supporting the tippler, and works freely in the slot formed by the guard, *g*. The projection on the catch-lever or pawl, L, engages with a square pin, *f*, while the tippler is being driven, and the end of the catch-lever, L, is lifted by the projection, *a*, as the tippler comes round. The gibbed lever, K, is prolonged downwards, and works between two stops which confine the movements of the belt to the width of the pulleys, the end of the lever being connected by a rod to the fork, M, controlling the strap.

When a tub of coal is pushed into the tippler (pushing the empty tub out before it), the hand-lever, H, is depressed until the pawl, L, engages with the pin, *f*, the gib being clear of the projection, *a*, on the tippler and the strap being moved from the loose, F, to the fast, F^1, pulley. The tippler then revolves, owing to the friction between the rollers, B and C, and the steel rings, A. On nearly completing a revolution, the pawl, L, is automatically released by the projecting plate, *a*; and the counter-poise, I, instantly causes the lever to move into a position that will stop the movement of the tippler, and the strap is moved on to the loose pulley, F.

7.—*The Heenan and Froude Rotary Tippler* (Figs. 18 and 19, Plate XV.).—This tippler is of the rotary power-driven type, having the cylindrical frame supported upon two pairs of rollers. The cast-iron ends, A, S, of the tippler are dissimilar, A having two grooves on the edge, so as to engage with the friction-wheel, B, when in motion. This friction-wheel is mounted on an arm, C, pivoted on the countershaft, D, from which motion is transmitted to the friction-wheel by the spur-wheels, E and F. The countershaft, D, is driven from the screen-engine, and is in constant motion. The arm, C, also carries a hand-lever, L, by which the friction-wheel is placed in contact with the grooved rim, A, of the tippler frame. An arm, G, with a counterweight attached, is used to throw the friction-wheel, B, out of gear, and to automatically stop the tippler at the

end of each revolution. The lever, H, having a small roller, *r*, and a pawl, *p*, mounted on one end, is pivoted loose on the shaft, D, and its movement is transmitted by means of the adjusting screws, *s* and s^1, to the arm C, and thence to the hand-lever, L, and friction-wheels. The screws, *s*, s^1, are so adjusted that when the roller, *r*, is out of the recess, *c*, and riding on the rim, A, the friction-wheel, B, is in gear.

After placing a tub on the tippler, the banksman pushes over the lever, L, so that the roller, *r*, and pawl, *p*, are lifted from the recess, *c*, on the rim, A, and the friction-wheel, B, is put into gear at one movement. On completing a revolution, the roller, *r*, drops into the recess, *c*, throwing the friction-wheel, B, out of contact with the rim, A, and the pawl, *p*, brings the tippler to rest in the proper position for receiving another tub.

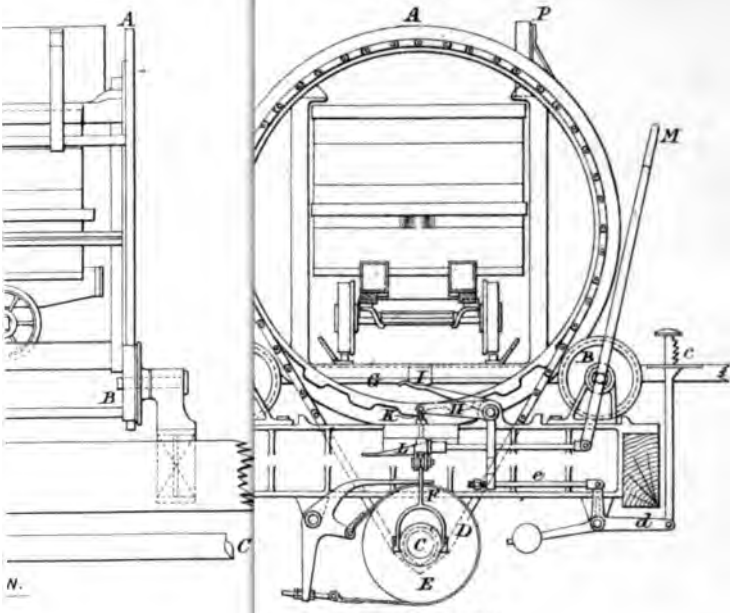
8.—*The Pelsall Automatic Side Tippler* (Figs. 20 and 21, Plate XV.).—This tippler consists of a pair of cast-iron sides secured to, and revolving on, a central shaft. Each of the three compartments is occupied by a tub, the weight of the loaded or partly loaded tubs, carrying the empty tubs back to the level of the pit-bank. The operation of tipping is regulated by means of a foot or hand-brake acting on the rim of the side frame.

9.—*The Tate Automatic Tippler and Weighing-machine* (Figs. 22, 23, and 24, Plate XV.).—This arrangement differs from those already described in being self-acting or automatic in its action, and combines an unique combination of necessary operations; (*a*) the number of tubs passed through the tippler is recorded on a counter, set at zero each morning; (*b*) the nett weight of the contents of each tub is ascertained; and (*c*) the contents are placed on the screen in the ordinary manner.

The tippler consists of a semi-circular closed vessel with flat ends, C C, supported under a central shaft, S, resting on bearings, G, fixed on an outside frame, B D. Upon the closed vessel, C, are riveted the rails for the tubs, and two pieces of angle-iron, A A, one on each side, to secure the tub in position whilst being emptied. The vessel, C, is filled with liquid and acts as a counter-balance to bring the empty tub and tippler back into its proper position after the tub has been teemed. The movement of the tippler through an angle of about 135 degrees, is regulated by means of an ordinary strap passing round the brake-wheel, W, keyed upon the end of the shaft, S.

The framework, B D, carrying the tippler, is suspended from, and connected to, the levers of an overhead self-recording weighing-machine

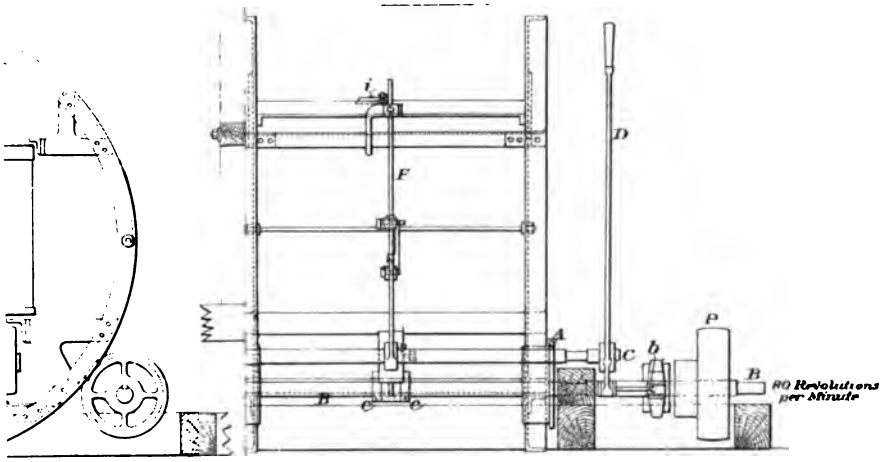
FIG. 7.



CROSS SECTION.

Scale 1 Foot to 3/8 Inch

FIG. 12.



SIDE VIEW.

COLLIERY COST-SHEETS.*

By J. J. PREST.

APPENDICES.

FORM A. .....COLLIERIES, ...PIT. ABSTRACT OF PAY BILLS AND SUNDRY INFORMATION. *Week ending*.....189 .

	IRONSTONE.						COAL.							
	Tons.	C.	Tons.	C.	Tons.	C.	Tons.	C.	Tons.	C.	Tons.	C.		
	Pennystone.		Gubbin.		Bassey Mine.		Bassey Mine.		Cannel Row.		Great Row.		Total.	Average Cost per Ton of Coal.
		Cost per Ton.		Cost per Ton.		Cost per Ton.		Cost per Ton.		Cost per Ton.		Cost per Ton.		
	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.
Underground—														
Getting														
Yard-work ...														
Allowance ...														
Totals ...														
Overmen ...														
Firemen ...														
Jigging														
Tramming ...														
Driving														
Door-keeping ...														
Engine-dips and horse-roads ...														
Enginemen and stokers ...														
Pumping water														
Hooking ...														
Horse-keeping ...														
Bricklayers ...														
Intake air-roads, repairs ...														
Return air-roads, repairs ...														
Totals ..														
Totals underground ...														
Bank—														
Banking-out ...														
Screening ...														
Teeming dirt ...														
Weighmen ..														
Winding and stoking ..														
Pumping ...														
Fan enginemen														
Lampmen ...														
Totals ...														

* *Trans. Fed. Inst.*, vol. viii., page 326.

FORM A.—*Continued.*

	Ironstone.						Coal.							
	Tons.	C.	Tons.	C.	Tons.	C.	Tons	C.	Tons.	C.	Tons.	C.	Total.	Average Cost per Ton of Coal.
	Pennystone.		Gubbin.		Bassey Mine.		Bassey Mine.		Cannel Row.		Great Row.			
		Cost per Ton.		Cost per Ton.		Cost per Ton.		Cost per Ton.		Cost per Ton.		Cost per Ton.		
	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.	£ s. d.	s. d.
Mechanics—														
Carpenters and tub-menders...														
Fitters														
Blacksmiths ...														
Bricklayers ...														
Cartage														
Labourage ...														
Shaftmen ...														
Sundries ...														
Totals ...														
Gross totals ..														

PRODUCTION.

Seams.	Coal.		Slack.		Black Band.		Clay Band.		Total Coal.		Total Iron-stone.		Total Production.		Dirt Picked from Coal, etc.		Dirt Drawn from Pit.	
	T.	Ct.	T.	Ct.	T.	Ct.	T.	Ct.	T.	Ct.	T.	Ct.	T.	Ct.	T.	Ct.	T.	Ct.
Pennystone... ...																		
Cannel Row ...																		
Great Row																		
Gubbin																		
Bassey Mine ...																		
Totals ...																		

SUNDRY INFORMATION.

Seams.	No. of Days Worked.	Tons Drawn in Night Shift.	Tons Drawn in Day Shift.	Total Tons Drawn.	Average Output per Day.	Shifts Worked by Colliers and Loaders.	Tons Worked per Man per Shift.	No. of Colliers and Loaders Employed.	Colliers' and Loaders' Average Earnings per Shift.	Long-wall.		Pillar and Stall.		Percentage of Men Employed.		Remarks.
														At Work.	Off Work.	
									s. d.	Tons.	%	Tons.	%			
Pennystone ...																
Cannel Row ...																
Great Row ...																
Gubbin ...																
Bassey Mine ...																
Totals ...																

FORM B. .....COLLIERY. GENERAL COST ACCOUNT FOR.....WEEKS ENDING.....189 .

	Days Worked.	Average per Week.	Tons Accounted for.	Coal in Stock.	Total.	Sales.	Colliery Consumption.	Remaining in Stock.
				Tons.	Tons.	Tons.	Tons.	Tons.
This month								
Previously								
Total								
Corresponding month last year								

	Amount.	Cost per Ton.	Cost per Ton Previous .... Weeks.	Remarks.
	£ s. d.	s. d.	s. d.	
Underground—				
Getting				
Yard-work				
Repairs				
Sundries				
Totals				
Surface—				
Screening				
Mechanics				
Enginemen and stokers ...				
Pumping				
Locomotives				
Railway				
Labourers				
Sundries				
Totals				
Total labour				
Materials—				
Timber and pit wood ...				
Iron and steel				
Grease and oils				
Bricks and mortar ...				
Ropes				
Provender				
Slack to boilers				
Sundries				
Totals				
General charges—				
Royalty rents				
Rates and taxes				
Salaries				
Sundries				
Totals				
Total cost per ton				
Average price realized ...				
Profit or loss				
Previous profit or loss ...				
Total profit or loss to date ...				

FORM C. .. ..COLLIERIES. SUMMARY OF COST SHEETS.

.....*Weeks ending*... .....189 .

PRODUCTION.					SALES AND CONSUMPTION.				RESULT.	
Name of Pit.	Description.	Tons.	Per Cent.	Average Cost per Ton.	Description.	Tons.	Per Cent.	Average Price Realized.	Profit.	Loss.
				s. d.				s. d.	s. d.	s. d.
	Coal				To Customers—					
	Cobbles ...				Coal					
	Nuts				Cobbles					
	Slack				Nuts					
	Burgy				Slack					
	Totals ...				Burgy					
	From stock ...				Totals ...					
	Totals ...				To Stock—					
	Cost per ton ...				Coal					
					Cobbles					
					Nuts					
					Slack					
					Burgy					
					Totals ...					
					To Works—					
					Coal					
					Cobbles					
					Nuts					
					Slack					
					Burgy					
					Totals ...					
	Cost per ton ..				Gross totals ...					
	Realized price per ton ...				Previously ..					
	Per ton ...				Total to date ...					

FORM D.—ABSTRACT OF LABOUR COSTS PER TON AT .....COLLIERY, FOR YEAR 189

This form consists of a series of columns, headed by the various items recorded in Form A, so that the year's operations are shown in a convenient form.

NORTH STAFFORDSHIRE INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD AT THE NORTH STAFFORD HOTEL, STOKE-UPON-TRENT, MAY 13TH, 1895.

MR. E. B. WAIN, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and confirmed.

COAL MINES REGULATION BILL, 1895.

The PRESIDENT read a paper by Mr. M. Walton Brown on the "Coal Mines Regulation Bill, 1895."

Mr. PREST questioned whether a controversial matter like the Bill dealt with by Mr. Brown should be discussed.

Mr. J. RICHARD HAINES (secretary) said the Bill could be discussed under the rules of The Federated Institution of Mining Engineers. He explained that the Bill had been re-modelled, so that it might be undesirable to discuss the measure until it was drafted as it was to be submitted to Parliament.

Mr. PREST thought that the consideration of the Bill was a matter which concerned mine owners and agents more than mining engineers.

The discussion was adjourned.

DISCUSSION UPON MR. A. HASSAM'S PAPER ON "GOB-FIRES IN LONGWALL WORKINGS."*

Mr. PREST said it was a singular circumstance that in the Hanley district particular seams were not so liable to gob-fires as the same seam in adjoining collieries. Could this be accounted for by physical differences in the nature of the coal or contiguous strata.

Mr. HAINES said he was afraid it was only a question of waiting long enough. He had been working the eight-feet cockshead coal for twenty-two years without a fire until six weeks ago. The same seam was liable to take fire in neighbouring collieries, and had done so three or four times in some of them.

Mr. W. N. ATKINSON asked whether it was not a fact that the seams specially liable to gob-fire in North Staffordshire—the cockshead and the bullhurst—were not worked in the district referred to by Mr. Prest?

Mr. HAINES said that was so.

Mr. PREST said Mr. Heath told him that the rowhurst seam was very liable to take fire.

Mr. J. C. CADMAN said that there was a great difference in the coal-seams of North Staffordshire; some were more liable to spontaneous combustion than others. The chief way to prevent spontaneous combustion was to get all the coal. Where there had been a gob-fire, according to his experience, it was through coal having been left.

Mr. GREGORY said that there was another seam liable to gob-fire in the Hanley district. He referred to the ten-feet coal-seam at Burslem, it had occasionally shown signs of heating, but no fire had been seen.

The PRESIDENT said that he did not know of any authentic case of solid coal taking fire in North Staffordshire.

Mr. HASSAM said that there had been twelve or fifteen cases in which solid coal had taken fire.

Mr. J. C. RASPASS said that at the colliery where he was engaged they were working four or five seams, but there was only one in which gob-fires occurred. It had been noticed that where they put new gob against old gob they were almost certain to have combustion, so that 3 feet along the whole distance was in one case packed with sand. Even in that case a

* *Trans. Fed. Inst.*, vol. viii., page 332.

gob-fire occurred. Between the roadways where the roof could weight properly on the gob, they were not much troubled with gob-fires. In order to deal with the matter more effectually, they had tried running a road 30 feet from the rib, so that the roof in the roadway could be kept in repair. It was important that all the timber should be removed.

The PRESIDENT asked whether there were any special characteristics in the seam rendering it liable to gob-fires?

Mr. HASSAM said that it was the better quality of coal—good house coal.

The PRESIDENT said he should have thought that where the coal was left over the packs, as in Mr. Hassam's case, it would have been less liable to cause trouble from heating than with ribs of coal left from roof to floor.

Mr. ARTHUR HASSAM said that he agreed with the President on that point.

The PRESIDENT thought that the remark was made at the former meeting that hardly any gob-fires occurred in that district except in seams where there was hussle (black carbonaceous shale), which formed a dense, close mass.

Mr. P. BRENNAN said that he had experience of a coal-seam, where there were no pillars, and there was no slack in the gob, and they could trace the gob-fire to the coal in the roof.

Mr. PREST asked whether any member had had experience of spontaneous ignition of coal stacked on the surface?

Mr. J. C. CADMAN said that he had known cases of that kind.

The PRESIDENT said that about two years ago he had a stack of coal on fire containing about 20,000 tons. The stack was about 20 feet high and was tipped as it came from the pit. Since then he had taken out the slack before stacking, and had had no further trouble with the same coal when stacking it on the surface.

Mr. W. N. ATKINSON said with respect to what had been stated as to fires in the solid coal, he thought that it was a misnomer. He had known one or two cases—one in North Staffordshire—where a fire had occurred in a pillar of coal, in the body of a pillar; but that was due to the coal being cracked, and the fire occurred in the fissure. He believed that was how fires, said to be in the solid, in South Staffordshire, really occurred. There was one point referred to by Mr. Raspass that he had never heard before, and that was that fires occurred where new gob was laid against old gob. Did the gob-fires never occur except where new goaf was placed against old goaf?

Mr. RASPASS replied that they did. They had occurred in ordinary workings, where the mine was in process of being worked for some time. But where they put new goaf against old goaf, they had had fires, most of them between the roadways and the ribs. They almost invariably occurred in that way.

The PRESIDENT asked whether that was possibly due to the pack between the roadway and the rib not being so solid and densely compressed as in the waste in the goaf, where there was a larger area? He was thinking of that in connexion with some heating he once saw in the eight feet coal-seam, where the coal was worked in wide stalls and narrow pillars were left on the Yorkshire system. When they were taken out the floor was so hot that he could scarcely put his hand on it.

Mr. DE RANCE said that in Lancashire, at Garswood Hall colliery, in the Wigan district, while sinking a shaft, a considerable quantity of petroleum was found, the dark oil being lifted in buckets. At a greater depth a large blower of gas was found, it was conveyed out of the shaft through a pipe and burnt with a large flame at the surface, and was visible for about 30 miles. At the same colliery, a large stack of Orrel four-feet coal was placed on the bank, and it took fire. He suggested that the presence of oil might produce spontaneous combustion?

The PRESIDENT said that he had heard of the case of a colliery where gob-fires were only found where oil had appeared, the oil being inflammable at 94 degs. Fahr.

Mr. DE RANCE said that he had found a dark-coloured shale in Cheshire. At Neston colliery, he took a piece which looked like oil-shale, and put it on his dressing-table, and thought nothing of it for some weeks. He then tried the effect of the sun's rays passing through a magnifying glass on this shale, and it speedily became heated and ignited.

Mr. J. C. CADMAN said that there had been fires in heaps of coal where there had been no oil.

The discussion was adjourned.

THE NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,
HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,
JUNE 8TH, 1895.

MR. THOMAS DOUGLAS, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting, and reported the proceedings of the Council at their meetings on May 25th and that day.

The proceedings of the Council of The Federated Institution of Mining Engineers were reported.

The SECRETARY read the Balloting List, as prepared by the Council, for the election of officers for the year 1895-96.

The following gentlemen were elected, having been previously nominated:—

MEMBERS—

Mr. DAVID ABRAHAM, Mining Engineer, Pencoedcae, Merthyr Tydvil.

Mr. ENRIQUE ALLCHURCH, Mining Engineer, Buenos Aires, Argentine Republic.

Mr. J. W. BATES, Mining Engineer, Surveyor, Estate Agent, and Valuer, Sutton Hall, St. Helens, Lancashire.

Mr. SIDNEY BATES, Manager, Mickley and Prudhoe Collieries, Mickley Colliery Offices, Stocksfield-upon-Tyne.

Mr. CUTHBERT BURNETT, Engineer, Fallowfield House, South Hetton.

Mr. W. H. CORBOULD, Mining Engineer, The Hannan's Reward Gold-mining Company, Limited, Hannan's Hill, Kalgoorlie, Western Australia.

Mr. FRITZ GILLMAN, Mining and Metallurgical Engineer, Gartenstrasse 1, Freiburg im/B, Baden, Germany.

Mr. JONATHAN HARRISON, Mining Engineer, Kimihia Colliery, Auckland, New Zealand.

Mr. WILLIAM PELLEW HARVEY, Mining Engineer and Assayer, Vancouver, British Columbia.

Mr. JOHN B. HASTINGS, Civil and Mining Engineer, Boise City, Idaho, U.S.A.

Mr. JACOB HODGSON, Mechanical Engineer, Cornsay Colliery, County Durham.

Mr. CARLOS A. LYNES HOSKOLD, Mining Engineer, Buenos Aires, Argentine Republic.

Mr. KWONG YUNG KWANG, Mining Engineer, Linsi Colliery, c/o Chinese Engineering and Mining Company, Tientsin, North China.

Mr. W. H. C. LOVELY, Mining Engineer, Metallurgist, Assayer, and Mine Surveyor, Coolgardie, Western Australia.

Mr. FRANCIS HERBERT MASON, Consulting Metallurgist and Assayer, Arlington Place, Truro, Nova Scotia.

Mr. WILLIAM JASPER NICOLLS, Mining and Civil Engineer, c/o Berwind-White Coal-mining Company, Betz Building, Philadelphia, Pa., U.S.A.

Mr. THOMAS GEORGE NOBLE, Manager, Sacriston Colliery, Durham.

Mr. CLAUD BOWES PALMER, Colliery Agent, Wardley Hall, Newcastle-upon-Tyne.

Mr. JOHN C. F. RANDOLPH, Mining Engineer and Metallurgist, Mill's Buildings, 35, Wall Street, and 15, Broad Street, New York, U.S.A.

Mr. NORMAN BACKHOUSE RIDLEY, Engineer, 58, Collingwood Street, Newcastle-upon-Tyne.

Mr. CHARLES M. ROLKER, Mining Engineer, Manager, Mining and Financial Trust Syndicate, Limited, 6, Drapers' Gardens, London, E.C.

Mr. JOHN SMAILES, Colliery Manager, Hebburn Colliery, Newcastle-upon-Tyne.

Mr. JOHN BARRET SQUIRE, Civil Engineer, 297, Clapham Road, London, S.W.

Mr. JOHN STANTON, Mining Engineer, 11 and 13, William Street, New York, U.S.A.

Mr. WILLIAM STEWART, Mining Engineer, Tillery Collieries, Abertillery, Monmouthshire.

Mr. DONALD M. D. STUART, Mining and Civil Engineer, Redland, Bristol.

Mr. FRANCIS WILLIAM THOMPSON, Civil and Mining Engineer, 15, Wood Street, Bolton, Lancashire.

Mr. JOHN G. THOMPSON, Assistant Manager, Bamfurlong, Wigan.

Mr. THOMAS A. WALKER, Engineer and Ironfounder, Pagefield Ironworks, Wigan.

Mr. HENRY WALL, Mining Engineer, King Street, Wigan.

Mr. HARRY BARTLETT WHITE, Mining Engineer and Surveyor, P.O. Box 24, Johannesburg, Transvaal, South African Republic.

Mr. HENRY WILCOX, Manager, Miners' Safety Explosive Works, Stanford-le-Hope, Essex.

ASSOCIATE MEMBERS—

Mr. WILLIAM HENRY FRANKLIN, Firth Park Villas, Sheffield.

Mr. JOHN E. ROGERSON, Oswald House, Durham.

Mr. THOMAS SCOWCROFT, Tonge Colliery, near Bolton.

Mr. E. GYBBON SPILSBURY, Managing Director, Trenton Iron Co., Trenton, New Jersey, U.S.A.

Mr. JOHN THOM, Canal Works, Patricroft, near Manchester.

Mr. OSWALD WALMESLEY, Barrister-at-Law, 2, Stone Buildings, Lincoln's Inn, London.

ASSOCIATES—

Mr. FRANK BOWMAN, Assistant Colliery Engineer, Auckland Park Colliery, near Bishop Auckland.
Mr. ALLAN BROWELL, Under-Manager, Cornsay Colliery, County Durham.
Mr. SAMUEL DUNNETT, Miner, Cambois Colliery, Northumberland.
Mr. H. W. HALBAUM, Stoneman, Boldon Colliery, R.S.O., County Durham.
Mr. JAMES HENRY, Under-Manager, Howgill Staith, Whitehaven,
Mr. WILLIAM ALLEN LOVE, Brickworks Foreman, Cornsay Colliery, County Durham.
Mr. WILLIAM MILLBURN, Surveyor, Cornsay House, Cornsay, Durham.
Mr. GEORGE ROSS PRATT, Under-Manager, Springwell Colliery, Gateshead.
Mr. JOHN ROWE, Under-Manager, Hollinside Terrace, Lanchester.
Mr. GEORGE TURNER, Deputy-Overman, Cornsay Colliery, County Durham.

STUDENTS—

Mr. MATTHEW WILLIAM ARCHER, Mining Student, West Stanley, R.S.O., County Durham.
Mr. WILLIAM MUSGROVE, Mining Student, Throckley Colliery, Newcastle-upon-Tyne.
Mr. JONATHAN SEVERS, Mining Student, Stanley, R.S.O., Newcastle-upon-Tyne.

The following paper on "Experiments with Explosives" by Mr. Winkhaus was read by the Secretary :—

EXPERIMENTS WITH EXPLOSIVES.

By BERGASSESSOR WINKHAUS.*

I.—The Experimental Gallery for Testing the Effect of Explosives on Fire-damp and Coal-dust at Schalke, Westphalia. †

The experimental gallery at the Consolidation pit (No. 1 shaft), near Schalke, was erected by the Westphalian Miners' Provident Society, of Bochum, with funds voted at the general meeting of March 2nd, 1894, and was in working order by the middle of the following August. One of the main reasons for selecting this pit as the *locus* of experiment was that it furnished an abundant source of natural pit-gas, which had for the last ten years been made use of in heating the compressed air, lighting up the shaft-bottom, and other similar purposes. Moreover, the site which was freely placed at the disposal of the experimenters by the owners of the mine was admirably adapted to the object in view.

The plan (Fig. 1, Plate XVI.) shows that the site is an area enclosed on two sides by two old pit-heaps, respectively 23 feet (7 metres) and 40 feet (12 metres) high, and is more than 650 feet (200 metres) distant from the nearest inhabited house. The experimental gallery is built into the north-western slope of the more southerly of the two pit-heaps, and is mostly modelled on a similar structure put up at the Royal König colliery at Neukirchen, in the Saarbrück coal-field.

The experimental gallery (Figs. 2, 3, and 4, Plate XVI.), arranged to accord as nearly as possible with the conditions which obtain underground, is 115 feet (35 metres) long and elliptical in section. Its inside height is 6·07 feet (1·85 metres) and its inside width is 4·43 feet (1·35 metres). The gallery is constructed of strong elliptically bent-frames made of H-iron, which are placed $15\frac{3}{4}$ to $25\frac{1}{2}$ inches (400 to 650 millimetres) apart, and lined with a triple covering of best pitchpine planks of a total thickness of 2·36 inches (60 millimetres). This lining of three layers of planks affords

* Translated by L. L. Belinfante, B.Sc.

† "Die berggewerkschaftliche Versuchsstrecke zur Ergründung der Einwirkung von Sprengstoffen auf Schlagwetter und Kohlenstaub auf Zeche Consolidation Schacht I bei Schalke in Westfalen." *Glückauf*, 1894, pages 1651-1654.

very great resisting power, and renders the sides of the gallery as nearly impermeable as possible. With the view of securing this last condition, the edges of the planks, which have a breadth of 6 inches (150 millimetres), are displaced by one-third of their breadth from each other in the separate layers (Fig. 3, Plate XVI.).

The gallery is open at one end, and at the other end is fixed into a block of masonry 9·84 feet (3 metres) broad, 12·30 feet (3·75 metres) high, and 9·84 feet (3 metres) long, which encloses the gallery, within by 31½ inches (80 centimetres), and without by 43¼ inches (1·10 metres). In order to secure as firm a connexion as possible between the gallery and the masonry-block, three stout angle-irons (*a*) are bolted to the exterior of the framework of the gallery, as well as three bars (*b*) to the first five iron rings, and their recurved ends are anchored 6·56 feet (2 metres) into the masonry.

In the gallery at the working-face represented by the masonry block, two crucible-steel cannons are embedded; these are of a pattern similar to that used at Saarbrück. They consist of a core 6½ inches (165 millimetres) in diameter over which is drawn, when hot, a shell 19½ inches (495 millimetres) in diameter. In this core is the hole from which the shots are fired, and this is 18·11 inches (460 millimetres) long, and 2·16 inches (55 millimetres) in diameter (Figs. 5 and 6, Plate XVI.). Ring-handles are attached to both cannons so as to facilitate manipulation in setting the cannons into and out of their required positions. The cannons are bedded in sand, at the middle and lower portion of the face, at an angle so that a line prolonging the axis of the shot-holes touches the roof of the gallery at a distance of about 32·81 feet (10 metres) from the face. At the back, the cannons are cushioned with sheet indiarubber 1·38 inches (35 millimetres) thick, and willow-wood boards 2 inches (50 millimetres) thick.

The openings in the roof of the gallery, which may be closed by the plugs (*c*), 8 inches in diameter, act as safety-valves. The same purpose is served by the manhole (*d*) in the vicinity of the working-face. It is oval in section, and is closed by an iron sliding-shutter, which, when experiments are made with fire-damp, is luted with plastic clay.

The explosion-chamber for experiments with fire-damp is placed at the face, a wooden ring (*e*) is built at a distance of 16·73 feet (5·10 metres) from the working-face, over which strong brown paper is stretched by means of an annular iron spring (which is in contact both with the wooden ring and with the walls of the gallery), and in this way a space of 353 cubic feet (10 cubic metres) is partitioned off.

For ventilating the gallery after a shot has been fired, a cast-iron pipe (*f*) of 9·84 inches (250 millimetres) internal diameter is cemented into the masonry-block; it enters immediately below the roof of the gallery, and is connected with a Pelzer ventilator (20 inches in diameter) driven by a small steam-engine. This pipe can be closed inside the gallery by means of a wooden plug which exactly fits the cone-shaped opening of the pipe, and is so arranged that it can be removed from the other end of the pipe. By this means, there is no need to enter the gallery in order to connect it with the ventilator after the firing of a shot. The process of ventilating the gallery does not occupy more than six or seven minutes, even where the most dense fumes have been produced.

Fifteen sight-holes (*g*), placed on one side of the gallery at about three-fourths of its height, enable the observer to note what is going on inside. These sight-holes (Figs. 7 and 8, Plate XVI.) or windows consist of strong coupled iron frames containing small glass panes 1 inch (25 millimetres) thick. They are made taut inwardly with sheet asbestos and outwardly with sheet indiarubber, 0·71 inch (18 millimetres) thick; these materials act as cushions and tend to prevent the panes from being broken. Near the masonry-block, the distances between the sight-holes average about 3·28 feet (1 metre), while towards the open end of the gallery the distances are increased up to 8·2 feet (2·5 metres).

Opposite the sight-holes, and at a distance of 49·2 feet (15 metres), stands the observation-building. Here, in the wall which faces the gallery, is a slit through which a single observer may overlook whole length of the gallery (Fig. 9, Plate XVI.).

The shots were electrically fired by a Bornhardt magneto-electric exploder placed in the observation-building. By means of two insulated copper wires, the exploder is connected with the two insulated copper knobs (*h*, h^1, Figs. 2 and 4, Plate XVI.), to which the ends of the conducting-wires from the electric detonators are connected.

The pit-gas used in the experiments was given off from an unworked seam G (rich in gas) of the Consolidation pit, belonging to the upper division of the bituminous coals. Between the 1,770 feet (540 metres) level and a level about 164 feet (50 metres) higher a rise-heading has been driven and a stopping inserted therein. In this stopping, a pipe is fixed and conveys the gas up the winding-shaft and over the pit-heap to the site of the experiments. The gas is stored in two cylindrical receivers (*b*) containing each about 212 cubic feet (6 cubic metres), and is drawn off as required by means of water-pressure. The arrangement

of the receivers and of the gas and water outlet and inlet-valves is shown in Fig. 10 (Plate XVI.), this is so contrived that the gas stored in one receiver may be drawn off whilst the other is being refilled. In order to determine exactly the amount of gas drawn off from the receivers, a gas-meter connected with the system of pipes is placed in the observation-building. The gas pipes enter at one-fifth of the height of the gallery at *m*, in the middle of the explosion-chamber (Figs. 2 and 4, Plate XVI.).

With the object of accelerating the diffusion of the gases (after their entry into the explosion-chamber) and their mixture with the atmospheric air, a fan consisting of two blades (*l*) is fixed to the roof of the gallery within the explosion-chamber at about three-quarters of its height: this can be set in rapid rotation by means of teethed-wheel gearing (*i*). By means of this simple fan, complete diffusion of the gases is effected in from four to five minutes, a result confirmed by the examination of samples of gas taken simultaneously at quarter, half, and three-quarters height of the gallery.

The fan is also used for inducing complete dispersion and a cyclonic whirling of the coal-dust within the explosion-chamber just before a shot is fired. For this purpose a pipe is placed in the roof of the gallery, opening into it just above the fan, and supports a vessel (*k*) containing about 122 cubic inches (2 litres). This vessel is filled with coal-dust, which is dropped directly on to the rapidly whirling fan. The wooden blades of the fan are protected from injury from the explosive blast by being placed, before the shot is fired, exactly in the line of the axis of the gallery, and, as a matter of fact, it is very rare that they have to be replaced.

This apparatus is, in its essentials, modelled upon that in use at the experimental gallery at the Wilhelm shaft of the Emperor Ferdinand Northern Railway Company at Polnisch-Ostrau.

II.—Experiments with Mining Explosives.*

The series of experiments made in the experimental gallery were designed to investigate the degree of security as against inflammation of fire-damp and coal-dust, afforded by the various explosives in use in the mining district of Dortmund. In an investigation of this description, the main object must evidently be to secure, under given conditions, results which are comparable one with the other. It does not matter so much of what kind these conditions are, so long as they are made as uniform as possible for all the various explosives in the several series of

* *Glückauf*, 1895, pages 559-562, 577-580, and 597-600.

experiments. But the possibility of making the shot-firing in the experimental gallery correspond exactly to the circumstances which obtain in practical mining work had to be considered as absolutely out of the question.

Nor can the degree of safety of a particular explosive be measured directly and solely from the results obtained with that explosive. One must, as a preliminary to forming a correct judgment, compare the whole of the results obtained with the several explosives, and bear in mind at the same time the experience which has been gained with them in dusty and fiery mines.

(1) *The Procedure of the Experiments.*

All the shots were fired without stemming from the borehole of the lower crucible-steel cannon (Figs. 2 and 3, Plate XVI.). In those cases, however, where the object was to observe the length of flame of an explosive, the upper cannon was used, because the opening of the borehole could be seen from the observation-building through the first window of the gallery.

The impalpable coal-dust used in the experiments, taken from the sweepings of the screening-apparatus of a bituminous coal-pit, was throughout obtained from the same source. Its uniformity was constantly checked by means of analyses and by comparative shots fired with a given quantity of an explosive whose ignition-limit was known. For this purpose, gelatine-dynamit was used in cartridges of 3·53 ounces (100 grammes) weight, it having previously been proved that this and no less quantity almost invariably ignited this coal-dust. The chemical analyses which, in common with all the other analyses mentioned in these pages, were carried out at the laboratory of the Westphalian Miners' Provident Society, in Bochum, showed that the composition of this coal-dust varied within very narrow limits, as proved by the analyses made on the following dates:—

	Sept. 19th, 1894.	Nov. 30th, 1894.	Dec. 27th. 1894.	Mar. 11th. 1895.	Mar. 22nd, 1895.
Coke	75	74·7	75·6	75·0	74·1
Volatile matter	23	22·6	22·7	22·0	23·0
Water	2	2·7	1·7	3·0	2·9
	100	100	100	100	100
Ash	5·5	5·3	5·3	5·1	6·3

Moreover the physical properties of the dust (fineness of state of division, etc.) were, as a consequence of its origin, uniform throughout.

In each experiment, a carefully-measured quantity of coal-dust, *i.e.*, 183 cubic inches (3 litres) was strewed over a length of 16·4 feet

(5 metres) of the gallery, from the inner end outwards, after the gallery had been carefully cleared of the dust remaining from the preceding experiment. Moreover, shortly before the shot was fired another charge of 122 cubic inches (2 litres) was scattered in the gallery by means of the fan.

When it was considered desirable to carry out experiments in explosive gaseous mixtures without the addition of coal-dust, the interior of the gallery was first of all carefully washed out by means of a jet of water under high pressure, and the same course was adopted whenever it was proposed to investigate the flames produced by the explosives. This precaution was found indispensable, because even the smallest accumulations of coal-dust in the crannies and crevices of the gallery-framework would more or less influence the appearances or the results observed.

The quantity of pit-gas introduced into the explosion-chamber of the gallery was measured by means of a 50 lights gas-meter placed in the observation-building. The time of inflow of the gases was made as uniform as possible, on the basis of about 5 cubic feet (150 litres) per minute. During the inflow of the gas the fan was made to revolve, and thus a continuous even mixture of air and gas was distributed throughout the explosion-chamber. By this means the superfluous air resulting from the inflow of gas, which escaped through rifts in the paper diaphragm shutting off the explosion-chamber, was always an uniform air-mixture, and was little, if at all, affected by the composition of the gaseous mixture present in the explosion-chamber. Samples of the gases were drawn off into testing-tubes of about 12 cubic inches (200 cubic centimetres) capacity by means of indiarubber piping passing through the roof of the gallery and having its aperture about the centre, and were in part submitted to chemical analysis.

These analyses showed that in cases where a small proportion of pit-gas was present, the content of pit-gas only varied slightly (0·10 to 0·05 and at most 0·20 per cent.) from the calculated percentage of marsh-gas, determined from the composition of the blower. This uniformity may be noted in the samples taken on the same day, as shown by the following table of samples:—

Date of Experiments. 1894.	Quantity of Gas introduced. Litres.	Cubic Ft.	Percentage of Marsh Gas.
September 25	800 ...	28·25 ...	5·80
	900 ...	31·78 ...	6·66
„ 28	900 ...	31·78 ...	7·10
	900 ...	31·78 ...	7·00
October 1	300 ...	10·59 ...	2·15
	300 ...	10·59 ...	2·10

Date of Experiments. 1894.	Quantity of Gas introduced. Litres.	Cubic Ft.	Percentage of Marsh Gas.
October 11	900	31·78	6·20
	900	31·78	6·30
„ 16	900	31·78	6·30
	900	31·78	6·20
„ 23	900	31·78	6·00
	300	10·59	2·10
November 5	300	10·59	2·31
	300	10·59	2·30
	300	10·59	2·34
1895.			
February 19	1,000	35·32	6·00
	1,000	35·32	6·20
	1,000	35·32	6·00
„ 22	1,100	38·85	7·00
	1,100	38·85	7·20
„ 28	1,100	38·85	6·90
	1,100	38·85	7·00
March 7	1,050	37·08	6·50
	1,050	37·08	6·70
„ 12	1,050	37·08	7·00
	1,050	37·08	7·00
	950	33·55	6·30
„ 15	1,050	37·08	6·50
	1,050	37·08	6·70
	1,050	37·08	6·70

The preceding table shows that great differences were observable only between samples taken on different days. It was therefore deemed unnecessary to analyse every sample, but care was taken to analyse, if possible, at least one sample on each day. The results yielded by that analysis were then taken as a criterion for the other samples.

Moreover, in experiments made with explosive gaseous mixtures, in order to obtain an immediate check, a further sample was drawn off in a Lohmann explosion-tube,* and the contents were ignited by means of the passage of an electric spark. With a little practice the proportion of marsh gas may be gauged to within ½ per cent., merely from the violence of the fire-damp explosion which takes place in the tube.

The great variation between samples taken on different days may fairly be attributed to the want of uniformity in the quantity of fire-damp present in the pit-gas. For example, this contained of marsh gas:—

1894, September 13	84 per cent.
„ February 21	75 „
1895, April 2	80 „

* *Zeitschrift für das Berg-, Hütten- und Salinen-Wesen im Preussischen Staate*, vol. xxxvii., page 87.

When it was desired to make experiments in the simultaneous presence of coal-dust and fire-damp mixtures, the dust was strewed along the floor of the gallery before the partitioning off of the explosion-chamber, and the addition of further coal-dust to the air was effected after the gases had been introduced, and a little before the shot was fired.

In order to take account of the possible influence of variations in temperature, the temperature of the interior of the gallery, as well as that of the coal-dust, was ascertained before every experiment. Unfortunately, it was impracticable to arrange absolutely uniform conditions of temperature, because the temperature of the gallery depended on whether in the course of the experiments ignition was more or less frequently produced. After the construction of a heating-apparatus* it was, however, found possible to regulate in some degree the temperature, so that it rarely fell below 59 degs. Fahr. (15 degs. Cent.), and rarely exceeded 68 degs. Fahr. (20 degs. Cent.). Indeed, within these limits, no difference as regards the results obtained was observable.

Finally, we may mention that it was sought to neutralize the effects of the weather (favourable or not) by experimenting as far as possible in close succession with explosives of similar character: for example, westfalit and dahmenit. The results thus obtained with the different explosives did not show that the weather had any marked effect.

(2) *The Programme of Experiments.*

In order to investigate as accurately as possible the properties of the various explosives, and in particular their behaviour in presence of fire-damp or coal-dust, or in mixtures of fire-damp and coal-dust, every explosive was subjected to the following series of experiments:—

1. Firing of a shot from the upper cannon in the absence of fire-damp and coal-dust, so as to observe the breadth and length of the flame.

2. Firing of a series of shots with varying weights of explosive in order to determine the smallest amount which could, under the following conditions, ignite the explosive gaseous mixture, provided that such ignition could be produced by the heaviest charges which it was possible to introduce into the cannon, say 18 to 21 ounces (500 to 600 grammes).

(*a*) In presence of coal-dust strewed in the gallery and suspended in the air, without any fire-damp being present.

* The heating-apparatus consists of two wrought-iron tubes 16·4 feet (5 metres) long, 2 inches (0·05 metre) in internal diameter, connected to the main steam-pipes. These tubes rest on strong iron chairs, about 2 feet from the bottom of the gallery, and are carried along the walls into the explosion-chamber. They can be supplied with steam as desired.

(*b*) In the midst of an explosive mixture containing 6 to 7 per cent. of fire-damp, without any coal-dust being present.

(*c*) In the midst of a mixture containing fire-damp, coal-dust being at the same time strewed in the gallery and in suspension in the air.

(*aa*) In presence of a gaseous mixture, such that the proportion of marsh gas present is just recognizable with a safety-lamp, say about 2½ per cent. of fire-damp.

(*bb*) In presence of an explosive gaseous mixture containing 6 to 7 per cent. of fire-damp.

With those explosives which, in presence of 2½ per cent. of fire-damp, did not, even with heavy charges of as much as 18 ounces (500 grammes), ignite the coal-dust, further experiments were made in presence of gaseous mixtures richer in fire-damp (say 5 per cent. of marsh gas), yet not in themselves explosive, coal-dust being strewed and in suspension as before.

(3) *The Explosives used in the Experiments.*

As the main object of the experiments was to investigate closely the behaviour of the so-called safety-explosives, the investigation as regards these was made as thorough as possible, while the explosives which are not ranged in the safety category were included in the series of experiments only so far as they might elucidate by comparison the conclusions to be arrived at concerning safety-explosives. Blasting-powder was altogether excluded, because the earlier investigations of the Prussian Fire-damp Commission had shown that it was highly dangerous in all fiery and dusty mines, and is therefore nowadays no longer allowed to be used in such mines.

The substances which came into purview were:—

I.—Among the non-safety explosives:—

1. Gelatine-dynamit.
2. Guhr-dynamit.
3. Gesteins-carbonit (stonite).*

II.—As safety-explosives:—

1. Wetter-dynamit, from the Schlebusch dynamite factory, consisting of† trinitro-glycerine, 52·9 per cent.; sulphate of magnesia (Epsom salts, $MgSO_4 + 7\ H_2O$), 32·7 per cent.; kieselguhr, 14·4 per cent.

* According to the manufacturer, this consists of nitro-glycerine, 68·0 per cent.; kieselguhr, 17·0 per cent.; nitrate of potassium, 10·5 per cent.; ryemeal, 3·5 per cent.; soot, 1·0 per cent.

† According to analyses made by the Director of the Provident Society's Laboratory (Dr. Broockmann of Bochum).

2. Carbonit (coal) which (according to the manager of the Schlebusch carbonite factory) consists of nitro-glycerine, 25·0 per cent.; nitrate of potassium, 34·0 per cent.; ryemeal, 38·5 per cent.; woodmeal, 1·0 per cent.; nitrate of barium, 1·0 per cent.; bicarbonate of sodium, 0·5 per cent. The chemical analysis of a sample showed that the proportion of nitro-glycerine was 29·5 per cent.

3. Sekurit from the Köln-Rottweil powder factories, consisting of* mono- and dinitro-benzol, 29 per cent.; nitrate of ammonium, 37 per cent.; nitrate of potassium, 34 per cent.

4. Roburit from the roburit factory of Witten on the Ruhr, consisting of* dinitro-benzole, 17·8 per cent.; nitrate of ammonium, 79·2 per cent.; chloride of ammonium and sulphate of ammonium, 0·3 per cent.; water (damp arising from long storage of the sample), 2·7 per cent.

5. Westfalit from the westfalit factory at Sinsen (belonging to the Westphalian-Anhalt Explosive Company), consisting of* nitrate of ammonium, 94·0 per cent.; resin, 5·4 per cent.; chloride of ammonium, 0·1 per cent.; sulphate of ammonium, 0·4 per cent.; impurities, 0·1 per cent.

6. Dahmenit from the Castrop Safety-explosive Company, consisting of* nitrate of ammonium, 93·3 per cent.; naphthalene, 4·8 per cent.; chlorate of potassium, 1·6 per cent.; chloride of ammonium, 0·1 per cent.; sulphate of ammonium, 0·2 per cent.

7. Progressit from the roburit factory at Witten on the Ruhr, consisting of* nitrate of ammonium, 89·1 per cent.; hydrochlorate of aniline, 4·7 per cent.; sulphate of ammonium, 6·0 per cent.; impurities, 0·2 per cent.

The samples of explosives which formed the material for investigation were invariably taken from the stores of various mines, the sekurit alone being obtained direct from the manufacturers, as there was no other available source of supply.

(4) *Preliminary Experiments.*

1. Preliminary experiments with gaseous mixtures containing fire-damp in order to test the uniformity of the mixtures made by the fan.

For this purpose a number of experiments were arranged with a mixture which was just barely explosive, that is, contained something like 6 per cent. of fire-damp. And in order to test the uniformity of this mixture, the ends of the copper-wires (used in the firing of the

* According to analyses made by the Director of the Provident Society's Laboratory (Dr. Broockmann of Bochum).

shots) were brought into the proximity of the aperture of the borehole in the lower cannon with their ends at a distance of about 0·20 inch (5 millimetres) apart. When thereupon an electric spark was made to jump across with the aid of the magneto-electric exploder, a feeble explosion of the fire-damp mixture followed regularly, proving that even at a spot most remote from the inflow of the gas an explosive mixture of gas was present, and that therefore the mixture was complete. Further, several samples were taken simultaneously from different heights in the gallery,* and they invariably yielded uniform results, as shown by the following example:—

Date of taking of Sample.	Sample from.	Duration of Running of Fan.	Quantity of Gas Used.		Results of Analyses, Fire-damp.
			Litres.	Cubic Ft.	Percentage.
1894.					
September 13 ...	¼ height				6·00
,, ...	½ ,,	6 minutes	800	28·25	6·00
,, ...	¾ ,,				6·00
,, 19 ...	¼ ,,				2·20
,, ...	½ ,,	3½ minutes	310	10·95	2·35
,, ...	¾ ,,				2·30

2. Experiments with detonators.

In the firing of the charges Bornhardt electric detonators alone were used. In order to test whether a single detonator was sufficient to ignite fire-damp mixtures, a number of detonators, as tabulated below, partly hanging free, partly tied together by half-dozens within the front portion of the borehole in the cannon, were caused to explode in gaseous mixtures containing from 6½ to 9 per cent. of marsh gas. No ignition of fire-damp ever took place:—

No. 3	detonator,	containing	8·3	grains,	or 0·54	grammes,	of fulminate of mercury.
No. 6	,,	,,	15·4	,,	1·00	,,	,, ,,
No. 7	,,	,,	23·1	,,	1·50	,,	,, ,,
No. 8	,,	,,	30·8	,,	2·00	,,	,, ,,

3. Experiments for determining the inflammability of the paper diaphragm, which was used to form an explosion-chamber at a distance of 16·73 feet (5·10 metres) from the face of the gallery.

As it did not seem at all impossible that the paper which formed the partition-wall of the explosion-chamber might be ignited by the flame from the shot-firing, and that thereby the observed results might be

* The samples were obtained by fixing to the roof of the gallery a rubber plug, in which three holes were bored. Through these holes were passed rubber tubes opening at respectively ¼, ½, and ¾ height above the floor of the gallery.

affected, a series of experiments were carried out with gelatine-dynamit and safety-explosives in the absence of coal-dust and in the presence of fire-damp mixtures, such as were not explosive in themselves, but might well cause a lengthening of the flame. It was proved that the paper partition set up at a distance of 16·73 feet (5·10 metres) from the face of the gallery could not possibly be set on fire, even though one used such heavy charges as 10·58 ounces (300 grammes) of safety-explosive and 5·29 ounces (150 grammes) of gelatine-dynamit in the presence of 4½ per cent. of fire-damp. It was therefore considered needless to concern oneself about a preliminary impregnation of the partition-wall with such substances as might protect it from ignition.

4. At the very beginning of the experiments it was found that with safety-explosives extremely diverse results were obtained, according as the charge of explosive reached up to the aperture of the borehole of the cannon or lay at the bottom of it, and according as a free space of varying length was left in front of the cartridge. In the first case the conditions appeared much the most dangerous. This applies, with cartridges of 1·18 to 1·38 inches (30 to 35 millimetres) diameter, even to charges of 14·11 ounces (400 grammes). Now, as it was impracticable with many explosives, and in particular with the plastic nitro-glycerine varieties, by filling the material into cartridges of larger diameter, to put these deep down into the borehole, and as it was found that with heavy charges the cartridges always reached right up to the aperture, a series of experiments was first carried out with every one of the explosives. In these experiments the cartridges were so set within the cannon that they exactly coincided with the foremost edge of the borehole. Precisely identical conditions were arranged for each explosive, and for any selected charge. Of course such experiments cannot be claimed as approaching in any way the conditions which actually obtain in the mine, but it was necessary to make them in order to get comparable results for such explosives as were fired in large quantities in the original cartridges. The results, however, enable one to see what quantity of an explosive, under the most unfavourable conditions, is capable of igniting an atmosphere in which fire-damp and coal-dust are present.

Another series of experiments was undertaken, wherein the explosive charges were simply placed on the bottom of the borehole, leaving in front of them a free space of about 6 inches (150 millimetres) in length. No stemming was used in these experiments, as in the previous series. The study of the effects of stemming was reserved for a special set of experiments.

(5) *Results of the First Series of Experiments.*

With regard to the first series of experiments, wherein the explosive was placed in the foremost portion of the cannon, we may remark:—

1. That flame-phenomena, more or less considerable, were observable in the case of every explosive. Those of greater intensity were noticed with gesteins-carbonit, guhr-dynamit, and gelatine-dynamit. The safety-explosives gave rise only to short flames, and with carbonit only a feeble flash of light was observed.

2. From the point of view of their behaviour in presence of coal-dust without fire-damp, gelatine-dynamit, sekurit, guhr-dynamit, and gesteins-carbonit are to be regarded as by far the most dangerous. Charges of about 3·53 to 4·41 ounces (100 to 125 grammes) almost invariably sufficed to ignite a dusty atmosphere.

Roburit* and wetter-dynamit proved much more reliable. With these explosives the coal-dust was ignited only by a charge of 10·58 to 12·35 ounces (300 to 350 grammes).

The explosives westfalit and dahmenit, the cartridge-cases of which had been strongly steeped in resin, paraffin, or ceresin, to protect them against damp, and were used, still in the original coverings, in ordinary working, ignited coal-dust as soon as the charge reached about 8·82 ounces (250 grammes). But if the explosive, instead of being enclosed in a paraffined covering was simply wrapped in an ordinary paper one, ignition of coal-dust did not take place, even with charges as heavy as 17·46 ounces (495 grammes).

With progressit the dangerous influence of the paraffined cartridge-cases was not at first noticed.† The biggest charges which could be placed in the cannon in the original cases (that is 15·88 ounces or 450 grammes) failed to ignite coal-dust. The same observation applies to carbonit.

3. In the presence of gaseous mixtures containing small percentages of fire-damp (say about 2¼ per cent. of marsh gas) the safety of the greater number of explosives was found to diminish in a very remarkable degree. The accompanying table shows the smallest quantity of the

* The roburit which formed the material under investigation was packed exclusively in so-called double cartridges. These consist of a paper case containing the explosive, which was enclosed in a second covering, strongly steeped in ceresin. The instructions issued by the manufacturers state that this second covering is to be removed before making use of the explosive, and this was accordingly done in carrying out the experiments.

† Compare, however, the later experiments with progressit.

several explosives which would produce ignition. Progressit and carbonit proved in this case also to be safe in charges of 14·11 to 15·88 ounces (400 to 450 grammes):—

Name of Explosive.	Weight of Explosive. Ounces.	Grammes.
Gelatine-dynamit	2·65 ...	75
Guhr-dynamit	2·65 ...	75
Gesteins-carbonit	3·92 ...	111
Sekurit	1·76 ...	50
Roburit	5·36 ...	152
Wetter-dynamit	7·05 ...	200
Westfalit*	10·58 ...	300
Dahmenit*	12·35 ...	350

4. In explosive gaseous mixtures containing high percentages of fire-damp (say 6½ to 7 per cent. of marsh gas) the following minimum charges of the various explosives sufficed to bring about ignition. No essential difference in regard to ignition was noticed when coal-dust was strewed at the same time, and such differences as were observable may be attributed to purely accidental circumstances:—

Name of Explosive.	Weight of Explosive.		Pit-gas without Coal-dust Percentage of Marsh Gas.	Weight of Explosive.		Pit-gas with Coal-dust. Percentage of Marsh Gas.
	Ounces.	Grams.		Ounces.	Grams.	
Gelatine-dynamit ...	1·76	50	6·20	1·58	45	6·50
Guhr-dynamit ...	2·05	58	6·50	1·06	30	6·80
Gesteins-carbonit ...	2·86	81	7·00	—	—	—
Sekurit	5·29	150	6·40	—	—	—
Wetter-dynamit ...	2·40	68	6·30	1·80	51	6·60
Roburit	5·43	154	6·10	4·59	130	5·80
Westfalit*	8·85	251	7·00	8·82	250	6·30
Dahmenit*	8·85	251	7·10	8·82	250	6·30
Progressit*	19·40	550	6·75	19·76	560	7·25

Carbonit, even in charges of 21·17 ounces (600 grammes), and in the presence of 7·3 per cent. of marsh gas, did not produce any explosion of fire-damp.

In the Appendix, the separate experiments are systematically arranged in accordance with the principles laid down in the programme of experiments.

(6) *Results of the Second Series of Experiments.*

In the second series of experiments, wherein the explosive was set down on the bottom of the borehole in the cannon, so that in front of it was left a free space of about 6 inches (150 millimetres) in length, the

* Repacked into non-paraffined cartridge-cases.

interesting result was obtained that this open borehole did not have a similarly favourable effect with all explosives. Every one of the safety-explosives, with the exception of carbonit, was experimented with. The last-named had to be excluded from this series, because the quantity of explosive in the original packages always filled the borehole up to the very end, and, as previously remarked, a repacking of the explosive into cartridges of larger diameter was held to be impracticable as well as dangerous on account of the contained nitro-glycerine. But on the other hand, a new explosive, namely, dahmenit A* was included in this second series of experiments.

Of non-safety explosives, gelatine-dynamit alone was experimented with.

On comparing Table A, sections 1 and 6, with Table B, sections 11 and 12 (Appendix), it will be seen that, in the case of gelatine-dynamit and roburit, no difference as to the quantity necessary for producing ignition of the explosive atmosphere could be observed between the two series of experiments. So that the open borehole was of no effect on these explosives at all events. Moreover, the chemical properties of guhr-dynamit, gesteins-carbonit, and sekurit lead one to infer that these also would have shown results coinciding with those obtained in the first series.

The other safety-explosives gave much more favourable results than on previous occasions, especially in the presence of coal-dust and gaseous mixtures containing small percentages of fire-damp. In explosive fire-damp mixtures, however, no essential difference was noted as compared with the first series.

With wetter-dynamit, even charges of 16·23 ounces (460 grammes) failed to produce ignition of coal-dust. Even in the presence of 2½ per cent. of fire-damp in addition, the ignition of the dust only took place with charges approaching 12·35 ounces (350 grammes).

Westfalit and dahmenit in the original paper cartridges showed no difference from the previous results.† But in cartridges which had been repacked, they both proved perfectly safe as regards coal-dust up to charges of 15·88 ounces (450 grammes), even in presence of 2½ per cent.

* Dr. Broockmann analysed a sample, which had the following composition:—Nitrate of ammonium, 91·4; naphthalene, 5·2; bichromate of potash, 2·6; water, chloride of ammonium, etc., 0·8.

† The apparently less favourable results with westfalit are in all probability due to the rather stronger impregnation with paraffin of the paper coverings of the cartridges.

of fire-damp. Ignition of the coal-dust only took place with about 5 per cent. of fire-damp and charges of 10·58 and 12·35 ounces (300 and 350 grammes) respectively.

Progressit behaved in what, at first sight, appeared to be an extraordinary fashion. Whereas in the first series of more critical experiments ignition was only produced with charges varying between 16·76 and 19·40 ounces (475 and 550 grammes), and in presence of gaseous mixtures containing a high percentage of fire-damp (bearing in mind also that the original paraffined coverings in which the explosive was enclosed tended to promote ignition)—now ignition of the coal-dust in suspension in the atmosphere was obtained almost invariably with only 12·35 ounces (350 grammes) of explosive and in the complete absence of fire-damp. These strangely discordant results, however, are solely attributable to an alteration in the chemical composition of the explosive. Thus the chemical analysis of a sample demonstrated that the proportion of hydrochlorate of aniline had been increased from 4·6 to 5·5 per cent., while that of sulphate of ammonium had been diminished from 6 to 2·3 per cent.* On the other hand, without the paraffined paper covering, the explosive showed once more a high degree of safety.

The newest of the explosives, dahmenit A, turned out to be a very safe one. Even paraffined-paper cartridge-cases seemed to do nothing towards helping ignition, and the charges had to be increased up to 19·76 or 20·11 ounces (560 or 570 grammes), and the presence of as much as 7 per cent. of fire-damp arranged in order to produce ignition. Smaller charges, even in so fiery a gaseous mixture and in paraffined cases, failed to ignite coal-dust.

Conclusions.

The curiously dissimilar behaviour of the various explosives in both series of experiments can only be explained somewhat as follows:—

The explosives which show in the second series results in nowise differing from those obtained in the first series of experiments, develop a flame of magnitude such that a free space of 6 to 7 inches (150 to 180 millimetres) in front of the explosive in the hole does not suffice to quench or deaden the flame sufficiently to prevent the ignition of the inflammable atmosphere. On the other hand, the protective effect of

* An analysis of samples showed nitrate of ammonium, 92·2; hydrochlorate of aniline 5·5; sulphate of ammonium, 2·3. It is therefore evident that the progressit which is put on the market varies considerably in chemical composition.

this free space must be very noticeable in the case of extremely short-flame safety-explosives, such as wetter-dynamit, westfalit, and dahmenit. With the last two, for instance, only in presence of 5 per cent. of fire-damp was the flame lengthened sufficiently to come into contact with the explosive atmosphere and ignite it. The experiments with inflammable gaseous mixtures show on the face of them that the extremely favourable results obtained with such explosives are solely attributable to this shortness of flame. For in these experiments, the fire-damp of course diffusing into the open borehole and the flame of the explosive finding within that free space an inflammable atmosphere ready to hand, an explosion was produced which travelled into the interior of the gallery, and so the results could not show any difference from those obtained in the first series of experiments. The correctness of this explanation may be further inferred on looking at the results of experiments Nos. 1 to 5 in Table C (Appendix). When even a very small quantity of coal-dust was strewed within the borehole in front of the explosive, ignition of the dust-laden atmosphere was once more obtained with westfalit and dahmenit in presence of 2¼ per cent. of fire-damp, despite the free space of about 7 inches (170 to 180 millimetres) in length, and this with charges which practically corresponded to those found necessary in the first series of experiments. In such cases an inflammable mixture was thus present inside the borehole within reach of the flame produced by the shot.

As to the special influence of impregnation of the paper cartridge-cases with paraffin or ceresin, the following explanation would probably apply:—

The temperature developed on detonation of the explosive is so high that the paraffin on the cartridge-cases is vapourized and ignited. In consequence thereof a longer and hotter flame is developed, which, even with small charges of explosive, is quite capable of igniting the coal-dust-laden atmosphere. This view appears to be confirmed by the experiments with progressit (of the original composition) and dahmenit A. Both explosives proved extremely safe as well in presence of fire-damp mixtures as in that of coal-dust. The temperatures developed on detonation of these explosives must be comparatively low, so low, in fact, that the paraffin on the paper cartridge-cases does not take fire. Thus it is evident that the paraffin has no unfavourable effect with these explosives.

An interesting observation may be noted with regard to progressit. In the experiments wherein this explosive showed so high a degree of safety, ignition was in no wise promoted by the paraffin on the paper

cartridge-cases. But so soon as the safety of the explosive was diminished by reason of its altered chemical composition, the temperature of detonation being therefore heightened, the influence of the original paper cartridge-cases was once more observable. Whether with stemmed shots that influence is equally damaging cannot be affirmed in the absence of further evidence. For this a more detailed investigation is required, and it will be carried out as soon as the cannon which has to be used for such investigations is provided with a deeper and narrower borehole. The results of this further investigation will be published shortly.

The detonating force, too, of an explosive is an important factor to bear in mind when considering the advantages or otherwise of using that explosive in a mine, and the experiments are to be supplemented in that direction also. These results will also be published as soon as they are completed.

APPENDIX.

TABLE A.—FIRST SERIES OF EXPERIMENTS.

REFERENCES: I, ignition; N, non-ignition.

1. GELATINE-DYNAMIT.*

No. of Experiment.	Date.	Explosive. Weight.		Explosive. Cartridge Cases.	Coal-dust.	Fire-damp.	Temperatures. Of the Gallery.		Temperatures. Of the Coal-dust.		Results.
	1894.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Aug. 9	1·80	51	Original.	Nil.	Nil.	59	15	..	..	*a*
2	,, 9	3·53	100	,,	With coal-dust.	,,	59	15	59	15	I
3	,, 9	2·65	75	,,	,,	,,	59	15	59	15	N
4	,, 9	3·56	101	,,	,,	,,	59	15	59	15	I
5	Sept. 21	3·56	101	,,	,,	,,	61	16	59	15	N
6	Oct. 31	1·76	50	,,	,,	2·20	46	8	52	11	N
7	,, 31	2·65	75	,,	,,	2·20	54	12	52	11	I
8	Aug. 16	1·58	45	,,	,,	6·50	..	..	..	..	I
9	Sept. 25	1·76	50	,,	Nil.	6·20	59	15	..	..	I

* Ignited with No. 3 detonator. *a* Light red flames, 12 feet (3·5 metres) long.

2. GUHR-DYNAMIT.*

	1894.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Sept. 20	2·47	70	Original.	Nil.	Nil.	59	15	..	..	*a*
2	,, 20	5·33	151	,,	With coal-dust.	,,	59	15	64	17½	I
3	,, 20	4·44	126	,,	,,	,,	60	15½	64	17½	N
4	,, 20	4·41	125	,,	,,	,,	60	15½	64	17½	I
5	,, 21	3·88	110	,,	,,	,,	60	15½	61	16	N
6	Nov. 12	2·82	80	,,	,,	2·25	52	11	52	11	I
7	,, 12	2·65	75	,,	,,	2·25	52	11	52	11	I
8	Sept. 22	1·76	50	,,	,,	6·20	59	15	59	15	I
9	,, 22	1·06	30	,,	,,	6·80	59	15	59	15	I
10	,, 25	2·05	58	,,	Nil.	6·10	52	11	..	..	I

* Ignited with No. 3 detonator. *a* Flames, 8·2 feet (2·5 metres long.

3. GESTEINS-CARBONIT (STONITE).*

No. of Experiment.	Date.	Explosive. Weight. Oz.	Grms.	Cartridge Cases.	Coal-dust.	Fire-damp. %	Temperatures. Of the Gallery. Fahr.	Cent.	Of the Coal-dust. Fahr.	Cent.	Results.
	1894.										
1	Oct. 20	1·41	40	Original.	Nil.	Nil.	45	7	..	..	*a*
2	,, 20	7·05	200	,,	With coal-dust.	,,	46	8	46	8	I
3	,, 20	3·67	104	,,	,,	,,	50	10	46	8	N
4	,, 20	5·33	151	,,	,,	,,	50	10	46	8	I
5	,, 20	4·66	132	,,	,,	,,	54	12	46	8	I
6	,, 20	4·34	123	,,	,,	,,	54	12	46	8	I
7	,, 20	2·12	60	,,	,,	2·25	52	11	46	8	N
8	,, 23	3·92	111	,,	,,	2·10	52	11	50	10	I
9	,, 26	2·86	81	,,	Nil.	7·00	52	11	..	..	I

* Ignited with No. 6 detonator. *a* Very bright flame, 13 feet (4 metres) long.

4. SEKURIT.*

	1894.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Sept. 29	3·53	100	Original.	Nil.	Nil.	46	8	..	..	*a*
2	,, 29	7·20	204	,,	With coal-dust.	,,	49	9½	49	9½	I
3	,, 29	5·29	150	,,	,,	,,	51	10½	49	9½	I
4	,, 29	3·53	100	,,	,,	,,	55	12½	49	9½	I
5	Oct. 1	3·53	100	Repacked in ordinary cartridge paper.	,,	,,	49	9½	50	10	I
6	,, 1	1·76	50	,, ,,	,,	,,	49	9½	50	10	N
7	,, 1	1·76	50	,, ,,	,,	2·10	50	10	50	10	I
8	,, 1	1·06	30	,, ,,	,,	2·10	55	13	50	10	N
9	,, 2	3·53	100	Original.	Nil.	6·50	50	10	..	..	N
10	,, 2	7·05	200	,,	,,	6·40	54	12	..	..	I
11	,, 3	5·29	150	,,	,,	6·40	52	11	..	..	I

* Ignited with No. 6 detonator. *a* Flame at the first window.

5. WETTER-DYNAMIT.*

	1894.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Sept. 19	2·72	77	Original.	Nil.	Nil.	55	12½	..	..	*a*
2	,, 19	11·29	320	,,	,,	,,	55	12½	..	..	*b*
3	,, 19	5·29	150	,,	With coal-dust.	,,	58	14½	64	17½	N
4	,, 19	8·82	250	,,	,,	,,	58	14½	64	17½	N
5	,, 19	12·07	342	,,	,,	,,	59	15	64	17½	I
6	,, 19	10·83	307	,,	,,	,,	61	16	64	17½	N
7	,, 19	11·50	326	,,	,,	,,	66	19	64	17½	N
8	,, 19	12·03	341	,,	,,	,,	63	17	66	19	N
9	,, 19	12·21	346	,,	,,	,,	63	17	66	19	N
10	,, 19	13·09	371	,,	,,	,,	63	17	66	19	I
11	,, 19	12·42	352	,,	,,	,,	61	16	..	..	N
12	,, 21	12·38	351	,,	,,	,,	61	16	64	17½	I
13	,, 21	10·69	303	,,	,,	,,	61	16	63	17	N
14	Oct. 11	3·81	108	,,	,,	2·10	62	16½	52	11	N
15	,, 31	5·71	162	,,	,,	2·20	54	12	54	12	N
16	Nov. 5	8·99	255	,,	,,	2·31	57	14	54	12	I
17	,, 5	7·05	200	,,	,,	2·31	57	14	54	12	I
18	,, 5	7·05	200	,,	,,	2·20	57	14	54	12	I
19	,, 5	5·47	155	,,	,,	2·25	59	15	54	12	N
20	,, 5	8·99	255	,,	,,	1·50	57	14	54	12	N
21	Oct. 16	1·90	54	,,	,,	6·30	46	8	50	10	N
22	,, 16	2·40	68	,,	,,	6·30	46	7½	50	10	I
23	Sept. 25	5·29	150	,,	Nil.	6·60	58	14½	..	..	I
24	,, 25	3·56	101	,,	,,	6·60	61	16	..	..	I
25	,, 25	1·80	51	,,	,,	6·60	64	18	..	..	I
26	Nov. 10	1·75	49½	,,	,,	7·20	46	7½	..	..	I

* Ignited with No. 3 detonator. *a* No flame observed. *b* Flame at first window.

6. ROBURIT.*

No. of Experiment.	Date.	Explosive. Weight. Oz.	Explosive. Weight. Grms.	Explosive. Cartridge Cases.	Coal-dust.	Fire-damp.	Temperatures. Of the Gallery. Fahr.	Temperatures. Of the Gallery. Cent.	Temperatures. Of the Coal-dust. Fahr.	Temperatures. Of the Coal-dust. Cent.	Results.
	1894.					%					
1	Aug. 11	2·82	80	Original.	Nil.	Nil.	57	14	..	..	*a*
2	,, 17	5·40	153	After taking off the outer covering.	With coal-dust.	,,	..	..	..	..	N
3	,, 17	11·64	330	,, ,,	,,	,,	..	..	..	..	I
4	Oct. 13	8·82	250	,, ,,	,,	,,	54	12	50	10	N
5	,, 13	10·58	300	Repacked in ordinary cartridge paper.	,,	,,	57	14	52	11	I
6	,, 13	9·70	275	,, ,,	,,	,,	59	15	52	11	N
7	,, 18	10·58	300	,, ,,	,,	,,	50	10	50	10	N
8	,, 18	10·58	300	,, ,,	,,	,,	50	10	50	10	I
9	,, 24	6·28	178	,, ,,	,,	2·40	46	8	46	8	I
10	,, 24	4·27	121	,, ,,	,,	2·40	52	11	50	10	N
11	,, 24	5·36	152	,, ,,	,,	2·40	52	11	50	10	I
12	,, 23	4·59	130	,, ,,	,,	5·80	43	6	50	10	I
13	,, 22	7·05	200	,, ,,	Nil.	6·10	48	9	..	..	I
14	,, 22	5·43	154	,, ,,	,,	6·10	50	10	..	..	I
15	,, 22	3·67	104	,, ,,	,,	6·10	54	12	..	..	N

* The explosives roburit, westfalit, and dahmenit were ignited with Nos. 6, 7, or 8 detonators, according as these were in a more or less fresh (unaltered) condition.

a Flame at the second window, 8 feet (2·5 metres) long.

7. WESTFALIT.*

	1894.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Sept. 26	3·53	100	Original.	Nil.	Nil.	59	15	..	..	*a*
2	Aug. 17	9·42	267	,,	With coal-dust.	,,	..	..	..	..	N
3	,, 17	11·75	333	,,	,,	,,	..	..	..	..	I
4	Sept. 26	10·41	295	,,	,,	,,	61	16	64	17½	I
5	,, 26	8·82	250	,,	,,	,,	64	18	64	17½	I
6	,, 26	7·09	201	,,	,,	,,	64	18	64	17½	N
7	Oct. 8	9·13	259	,,	,,	,,	58	14½	59	15	I
8	,, 8	9·56	271	Repacked in non-paraffined paper.	,,	,,	58	14½	59	15	N
9	,, 8	14·11	400	,, ,,	,,	,,	61	16	59	15	N
10	Sept. 26	17·50	496	,, ,,	,,	,,	64	17½	64	17½	N
11	Nov. 3	16·76	475	,, ,,	,,	,,	..	..	..	..	N
12	Oct. 9	8·82	250	,, ,,	,,	2·10	46	8	48	9	N
13	,, 9	10·58	300	,, ,,	,,	2·10	46	8	54	12	I
14	Nov. 2	11·41	325	,, ,,	,,	2·20	55	13	54	12	N
15	,, 5	12·38	351	,, ,,	,,	2·34	57	14	54	12	I
16	Dec. 5	12·49	354	,, ,,	,,	2·25	57	14	59	15	N
17	,, 5	12·56	356	,, ,,	,,	2·25	57	14	59	15	I
18	,, 5	12·63	358	,, ,,	,,	2·25	59	15	59	15	I
19	Oct. 11	7·05	200	,, ,,	,,	6·20	52	11	50	10	N
20	,, 11	8·82	250	,, ,,	,,	6·20	52	11	55	12½	I
21	Sept. 28	7·05	200	,, ,,	Nil.	7·00	52	11	..	..	N
22	,, 28	8·85	251	,, ,,	,,	7·00	58	14½	..	..	I
23	Oct. 9	8·82	250	,, ,,	,,	6·30	60	15½	..	..	N
24	,, 9	10·58	300	,, ,,	,,	6·30	46	7½	..	..	I
25	,, 11	8·82	250	,, ,,	,,	6·30	47	8½	..	..	I
26	,, 8	7·05	200	Original.	,,	6·30	60	15½	..	..	I
27	,, 8	5·29	150	,,	,,	6·30	61	16	..	..	N

a Flame at the first window.

8. DAHMENIT.*

No. of Experiment.	Datt.	Explosive. Weight. Oz.	Grms.	Cartridge Cases.	Coal-dust.	Fire-damp. %	Temperatures. Of the Gallery. Fahr.	Cent.	Of the Coal-dust. Fahr.	Cent.	Results.
	1894.										
1	Sept. 26	3·53	100	Original.	Nil.	Nil.	59	15	..	..	*a*
2	,, 26	10·37	294	,,	With coal-dust.	,,	59	15	64	17½	I
3	,, 26	8·82	250	,,	,,	,,	64	17½	64	17½	I
4	,, 26	7·09	201	,,	,,	,,	64	18	63	17	N
5	,, 26	12·42	352	Repacked in non-paraffined paper.	,,	,,	64	17½	63	17	N
6	,, 26	17·50	496	,, ,,	,,	,,	64	18	63	17	N
7	Oct. 1	12·70	360	,, ,,	,,	2·15	55	12¾	50	10	I
8	Dec. 4	12·35	350	,, ,,	,,	2·25	54	12	57	14	N
9	,, 4	12·56	356	,, ,,	,,	2·25	57	14	59	15	I
10	,, 4	12·66	359	,, ,,	,,	2·25	66	19	59	15	I
11	,, 4	12·49	354	,, ,,	,,	2·27	65	18½	59	15	I
12	,, 4	10·62	301	,, ,,	,,	2·25	64	18	59	15	N
13	Oct. 11	8·82	250	,, ,,	,,	6·30	60	15½	52	11	I
14	,, 11	7·05	200	,, ,,	,,	6·30	64	17½	52	11	N
15	Sept. 27	5·29	150	,, ,,	Nil.	6·50	57	14	..	..	N
16	,, 27	7·05	200	,, ,,	,,	7·00	52	11	..	..	N
17	,, 27	8·85	251	,, ,,	,,	7·10	57	14	..	..	I
18	Oct. 5	9·70	275	,, ,,	,,	6·40	52	11	..	..	N

* The explosives roburit, westfalit, and dahmenit were ignited with Nos. 6, 7, or 8 detonators, according as these were in a more or less fresh (unaltered) condition.

a Flame at the first window.

9. PROGRESSIT.*

	1894.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Oct. 29	15·52	440	Original.	With coal dust.	Nil.	48	9	50	10	N
2	,, 5	14·11	400	,,	,,	2·25	54	12	52	11	N
3	,, 29	15·52	440	,,	,,	6·50	54	12	52	11	N
4	,, 29	15·52	440	,,	,,	7·25	54	12	52	11	N
5	,, 31	19·23	545	Repacked in non-paraffined paper.	,,	6·50	59	15	54	12	N
6	,, 31	19·76	560	,, ,,	,,	7·25	57	14	54	12	I
7	Nov. 8	16·23	460	Original.	,,	6·50	46	8	50	10	N
8	,, 8	16·76	475	,,	,,	6·50	46	8	54	12	I*a*
9	,, 9	16·93	480	,,	,,	6·50	48	9	52	11	N
10	,, 30	15·91	451	,,	,,	6·50	68	20	57	14	N
11	,, 30	16·55	469	,,	,,	6·50	68	20	57	14	N
12	Oct. 5	12·35	350	,,	Nil.	6·50	54	12	..	..	N
13	,, 29	15·52	440	,,	,,	7·25	50	10	..	..	N
14	Dec. 18	19·40	550	,,	,,	6·75	54	12	..	..	I

* Ignited with No. 8 detonator.

a Another coal-dust, containing 21·2 per cent. of volatile matter.

10. KOHLEN-CARBONIT.*

No. of Experiment.	Date.	Explosive. Weight.		Explosive. Cartridge Cases.	Coal-dust.	Fire-damp.	Temperatures. Of the Gallery.		Temperatures. Of the Coal-dust.		Results.
	1894.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Oct. 18	2·19	62	Original.	Nil.	Nil.	..	..	..	..	*a*
	1895.										
2	Mar. 20	7·41	210	,,	,,	,,	..	..	..	..	*b*
	1894.										
3	Oct. 18	7·05	200	,,	With coal-dust.	,,	42	5½	50	10	N
4	,, 18	9·03	256	,,	,,	,,	45	7	50	10	N
5	,, 18	10·72	304	,,	,,	,,	45	7	50	10	N
6	,, 18	12·35	350	,,	,,	,,	52	11	50	10	N
7	,, 20	15·70	445	,,	,,	,,	52	11	46	8	N
8	,, 20	10·55	299	,,	,,	2·25	52	11	46	8	N
9	,, 24	15·73	446	,,	,,	2·25	57	14	50	10	N
10	,, 24	15·66	444	,,	,,	6·50	54	12	50	10	N
11	,, 26	15·80	448	,,	,,	7·25	54	12	50	10	N
12	Nov. 29	15·66	444	,,	,,	6·50	73	23	57	14	N
13	,, 29	18·13	514	,,	,,	6·50	81	27	57	14	N
14	,, 29	18·10	513	,,	,,	7·30	84	29	68	20	N
	1895.										
15	Jan. 24	17·46	495	,,	,,	6·50	68	20	61	16	N
16	,, 26	18·13	514	,,	,,	6·75	64	18	51	12	N
17	,, 26	17·46	495	,,	,,	7·25	68	20	50	10	N
18	,, 26	21·38	606	,,	,,	7·25	77	25	57	14	N
	1894.										
19	Oct. 23	12·35	350	,,	Nil.	6·50	52	11	..	..	N
20	,, 23	15·88	450	,,	,,	6·50	52	11	..	..	N
21	Nov. 29	15·77	447	,,	,,	6·50	68	20	..	..	N
	1895.										
22	Mar. 26	16·41	465	,,	,,	6·50	63	17	..	..	N
23	,, 26	17·08	484	,,	,,	7·00	68	20	..	..	N
24	,, 26	21·41	607	,,	,,	7·00	66	19	..	..	N
25	,, 26	21·17	600	,,	,,	7·30	66	19	..	..	N

* Ignited with No. 6 detonator.

a No flame. *b* Very feeble shimmer of light at the first window.

TABLE B.—SECOND SERIES OF EXPERIMENTS.

REFERENCES: I, ignition; N, non-ignition.

11. GELATINE-DYNAMIT.

No. of Experiment.	Date.	Explosive. Weight.		Explosive. Cartridge Cases.	Coal dust.	Fire damp.	Temperatures. Of the Gallery.		Temperatures. Of the Coal-dust.		Results.
	1895.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Jan. 15	3·84	109	Original.	With coal dust.	Nil.	50	10	46	8	I
2	,, 19	3·67	104	,,	,,	,,	50	10	50	10	I
3	Mar 18	3·53	100	,,	,,	,,	59	15	59	15	I
4	,, 19	3·63	103	,,	,,	,,	43	6	59	15	I
5	,, 19	2·65	75	,,	,,	2·25	50	10	59	15	I
6	April 20	2·65	75	,,	,,	1·60	59	15	59	15	I

12. ROBURIT.

No. of Experiment.	Date.	Explosive. Weight. Oz.	Grms.	Explosive. Cartridge Cases.	Coal-dust.	Fire-damp. %	Temperatures. Of the Gallery. Fahr.	Cent.	Of the Coal-dust. Fahr.	Cent.	Results.
1	1895. Jan. 5	10·58	300	Repacked in non-paraffined paper.	With coal-dust.	Nil.	54	12	41	5	I
2	,, 5	9·03	256	,, ,,	,,	,,	50	10	41	5	N
3	,, 5	8·99	255	,, ,,	,,	,,	64	18	41	5	N
4	,, 5	10·76	305	,, ,,	,,	,,	46	8	41	5	I
5	,, 5	7·13	202	,, ,,	,,	2·25	70	21	41	5	I
6	,, 5	5·36	152	,, ,,	,,	2·25	64	18	41	5	I
7	,, 8	5·43	154	,, ,,	Nil.	6·50	63	17	..	..	I
8	,, 8	5·43	154	,, ,,	,,	6·50	61	16	..	..	I
9	,, 8	4·30	122	,, ,,	,,	6·50	61	16	..	..	I

13. WETTER-DYNAMIT.

	1895.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Jan. 15	14·43	409	Original.	With coal-dust.	Nil.	46	8	46	8	N
2	Mar. 15	14·22	403	,,	,,	,,	59	15	59	15	N
3	,, 26	16·30	462	,,	,,	,,	61	16	59	15	N
4	Jan. 15	10·41	295	,,	,,	2·25	50	10	46	8	N
5	,, 18	12·98	368	,,	,,	2·25	54	12	57	14	I
6	,, 19	10·62	301	,,	,,	2·25	54	12	50	10	N
7	Mar. 18	8·82	250	,,	,,	2·30	64	18	59	15	N
8	,, 18	10·83	307	,,	,,	2·30	64	18	59	15	N
9	,, 18	12·59	357	,,	,,	2·30	70	21	59	15	I
10	,, 28	13·93	395	,,	,,	1·30	63	17	59	15	I
11	Jan. 15	3·60	102	,,	,,	6·50	50	10	46	8	I
12	,, 15	2·54	72	,,	,,	6·50	55	13	46	8	N
13	Mar. 15	2·79	79	,,	,,	6·50	68	20	59	15	I
14	,, 15	2·63	76	,,	Nil.	6·50	68	20	..	..	I

14. WESTFALIT.

	1895.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Jan. 23	8·47	240	Original.	With coal-dust.	Nil.	57	14	50	10	I
2	,, 23	6·14	174	,,	,,	,,	61	16	54	12	I
3	Mar. 19	7·05	200	,,	,,	,,	57	14	59	15	I
4	,, 19	17·64	500	Repacked in non-paraffined paper.	,,	,,	57	14	59	15	N
5	Jan. 22	19·23	545	,, ,,	,,	2·25	64	18	46	8	N
6	,, 22	23·04	653	,, ,,	,,	2·25	66	19	50	10	N
7	,, 12	15·98	453	,, ,,	,,	3·50	61	16	54	12	N
8	,, 12	15·91	451	,, ,,	,,	3·50	64	18	54	12	N
9	,, 12	15·98	453	,, ,,	,,	4·25	61	16	54	12	N
10	1894. Dec. 31	12·52	355	,, ,,	,,	5·00	54	12	59	15	I
11	,, 31	10·83	307	,, ,,	,,	5·00	59	15	61	16	I
12	,, 31	8·82	250	,, ,,	,,	5·00	63	17	63	17	N
13	1895. Jan. 9	10·55	299	,, ,,	,,	5·00	46	8	45	7	I
14	,, 9	9·10	258	,, ,,	,,	5·00	64	18	61	16	N
15	1894. Dec. 28	9·13	259	,, ,,	,,	6·50	59	15	54	12	I
16	1895. Mar. 15	9·10	258	,, ,,	,,	6·50	68	20	59	15	N
17	1894. Dec. 27	11·40	323	,, ,,	Nil.	6·50	61	16	..	..	I
18	,, 27	10·65	302	,, ,,	,,	6·50	59	15	..	..	I
19	1895. Mar. 12	9·28	263	,, ,,	,,	7·00	68	20	..	..	N
20	,, 12	10·80	306	,, ,,	,,	7·00	68	20	..	..	I
21	,, 12	9·10	258	,, ,,	,	7·00	66	19	..	..	I
22	,, 12	8·96	254	,, ,,	,,	6·30	70	21	..	..	I

15. DAHMENIT.

No. of Experiment.	Date.	Explosive. Weight.	Explosive. Cartridge Cases.	Coal-dust.	Fire-damp.	Temperatures. Of the Gallery.		Temperatures. Of the Coal-dust.		Results.
	1895.	Oz. Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Mar. 6	8·54 242	Original.	With coal-dust.	Nil.	54	12	59	15	N
2	,, 6	12·66 359	,,	,,	,,	54	12	59	15	I
3	,, 6	10·58 300	,,	,,	,,	54	12	59	15	N
4	,, 26	10·23 290	,,	,,	,,	54	12	59	15	I
5	,, 6	16·09 456	Repacked in non-paraffined paper.	,,	2·30	68	20	59	15	N
6	,, 7	16·05 455	,, ,,	,,	2·25	68	20	59	15	N
7	Jan. 11	15·98 453	,, ,,	,,	3·50	61	16	54	12	N
8	,, 11	10·58 300	,, ,,	,,	5·00	59	15	46	8	N
9	,, 11	10·86 308	,, ,,	,,	5·00	57	14	50	10	N
10	,, 11	12·63 358	,, ,,	,,	5·00	59	15	57	14	I
11	,, 11	10·94 310	,, ,,	,,	5·00	59	15	57	14	N
12	Mar. 15	8·99 255	,, ,,	,,	6·60	70	21	59	15	N
13	,, 15	9·06 257	,, ,,	,,	6·70	68	20	59	15	N
14	,, 15	11·01 312	,, ,,	,,	6·70	66	19	59	15	N
15	,, 26	10·58 300	,, ,,	,,	6·90	68	20	59	15	I
16	,, 12	10·83 307	,, ,,	Nil.	7·00	68	20	..	..	I
17	,, 12	9·06 257	,, ,,	,,	7·00	68	20	..	..	N
18	,, 12	9·06 257	,, ,,	,,	7·00	68	20	..	..	I

16. PROGRESSIT.

	1895.	Oz. Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Feb. 28	7·76 220	Original.	Nil.	Nil.	64	18	..	..	*a*
2	,, 28	11·75 333	,,	With coal-dust.	,,	64	18	57	14	I
3	,, 28	9·17 260	,,	,,	,,	64	18	57	14	N
4	,, 28	12·31 349	,,	,,	,,	61	16	59	15	N
5	Mar. 2	12·07 342	,,	,,	,,	64	18	59	15	I
6	,, 8	12·13 344	,,	,,	,,	54	12	59	15	N
7	,, 8	12·91 366	,,	,,	,,	59	15	50	10	I
8	,, 8	12·10 343	,,	,,	2·25	68	20	50	10	I
9	,, 8	8·82 250	,,	,,	2·25	70	21	50	10	N
10	,, 8	10·65 302	,,	,,	2·25	68	20	50	10	I
11	Feb. 28	14·57 413	Repacked in non-paraffined paper.	,,	5·00	66	19	57	14	N
12	,, 26	17·57 498	,, ,,	,,	7·00	66	19	59	15	I
13	,, 26	16·23 460	,, ,,	,,	7·00	66	19	59	15	I
14	Mar. 2	9·56 271	,, ,,	,,	7·00	70	21	59	15	N
15	,, 2	13·44 381	,, ,,	,,	7·00	70	21	59	15	I
16	,, 2	14·18 402	,, ,,	,,	6·20	64	18	59	15	N

a Flame at the first window.

17. DAHMENIT A.*

	1895.	Oz. Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Mar. 4	8·82 250	Original.	Nil.	Nil.	57	14	..	..	*a*
2	Feb. 22	14·18 402	,,	With coal-dust.	,,	59	15	57	14	N
3	April 5	17·71 502	,,	,,	,,	50	10	59	15	N‡
4	,, 5	17·85 506	,,	,,	2·25	63	17	59	15	N‡
5	Feb. 22	14·22 403	,,	,,	5·75	72	22	59	15	N
6	April 5	17·75 503	,,	,,	5·00	63	17	59	15	N‡
7	Feb. 22	15·84 449	,,	,,	7·00	72	22	61	16	N
8	,, 22	20·43 579	Repacked in non-paraffined paper.†	,,	7·20	72	22	59	15	I
9	April 5	17·67 501	Original.	,,	6·30	66	19	59	15	N‡
10	,, 23	17·85 506	,,	Nil.	6·75	57	14	59	15	N‡
11	,, 5	17·64 500	,,	,,	6·30	68	20	..	..	N‡
12	,, 5	19·40 550	Repacked in non-paraffined paper.†	,,	6·30	66	19	..	..	N
13	,, 5	19·83 562	,, ,,	,,	6·30	61	16	..	..	N
14	,, 5	19·83 562	,, ,,	,,	7·00	66	19	..	..	I
15	,, 23	17·64 500	Original.	With coal-dust.	6·75	59	15	59	15	N

a Flame at the first window. * Ignited with No. 7 detonator. † Explosive was repacked, in order to use large quantities. ‡ The cartridges reached to the aperture of the borehole.

TABLE C.—THIRD SERIES OF EXPERIMENTS.

REFERENCES: I, ignition; N, non-ignition.

About ¼ ounce (15 grammes) of coal-dust was uniformly distributed within the borehole of the cannon. The explosive was set on the bottom of the borehole, a free space being left, about 7 inches (between 170 and 180 millimetres) long.

18. WESTFALIT.

No. of Experiment.	Date.	Explosive.			Coal-dust.	Fire-damp.	Temperatures.				Results.
		Weight.		Cartridge Cases.			Of the Gallery.		Of the Coal-dust.		
	1895.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
1	Jan. 22	15·98	453	Repacked in non-paraffined paper.	With coal-dust.	2·25	68	20	50	10	I
2	,, 22	13·72	389	,, ,,	,,	2·25	72	22	59	15	I
3	April 9	13·90	394	,, ,,	,,	2·20	68	20	59	15	I

19. DAHMENIT.

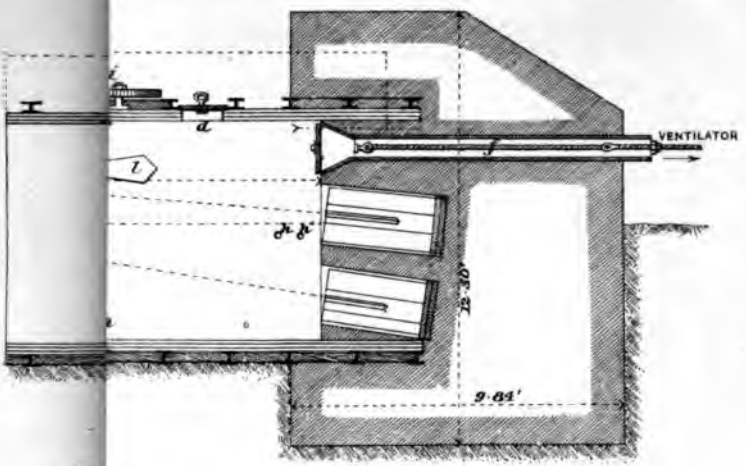
	1895.	Oz.	Grms.			%	Fahr.	Cent.	Fahr.	Cent.	
4	Mar. 26	14·11	400	Repacked in non-paraffined paper.	With coal-dust.	2·25	68	20	59	15	I
5	,, 26	13·30	377	,, ,,	,,	2·25	66	19	59	15	I

DISCUSSION UPON THE "REPORT OF THE PROCEEDINGS OF THE EXPLOSIVES' COMMITTEE"* AND UPON MR. WINKHAUS' "EXPERIMENTS WITH EXPLOSIVES."†

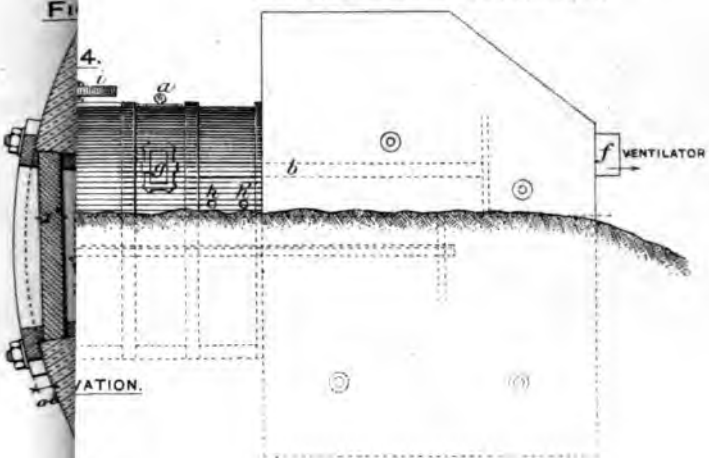
Mr. W. J. ORSMAN (Wigan) said that Mr. Winkhaus' paper could not be placed side by side with the *Report* of the Committee of The North of England Institute of Mining and Mechanical Engineers, because all the explosives described in the paper were "made in Germany" and were not in anyway similar (the composition being totally different) with the explosives made in England under practically the same names. With regard to Mr. Winkhaus' conclusions, it was stated that different results had been obtained on different occasions from the different explosives, and the result was attributed to shortness of flame. He (Mr. Orsman) had lately obtained from the different makers samples of each of the explosives mentioned in the paper, and he found in every case where the explosives had been mentioned as not igniting the gas it was really due, not to any apparent goodness of the explosive itself, but to imperfect

* *Trans. Fed. Inst.*, vol. viii., pages 227 and 593, and vol. ix., pages 115 and 206.

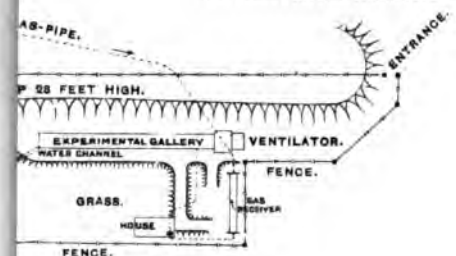
† *Ibid.*, vol. ix., page 250.



Fi...

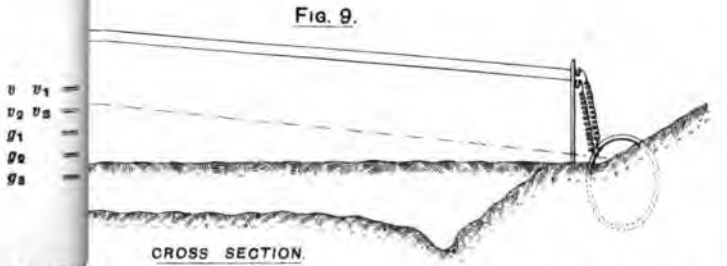


...VATION.



...OUND PLAN.

Fig. 9.



CROSS SECTION.

detonation. If those who had followed carefully the composition of explosives would take the trouble to look into the composition of the explosives mentioned as not igniting explosive mixtures of gas, they would find the composition was such as to yield, on detonation, a certain proportion of carbon monoxide, or in other words, incomplete combustion took place, either owing to a deficiency of oxygen or to the detonator not being strong enough to detonate all the explosive. In almost every case where incomplete combustion was obtained, there would be non-ignition of gas, consequently the experiments in the paper were very incomplete, as it was not stated whether each cartridge had been wholly detonated. It was a very ingenious argument, to state that when the explosive did fire the gas it was due to the paraffin on the paper cover, but it was not perfectly correct, for one could easily see that if the ceresin or paraffin did happen to be heated very strongly by the explosive, if there was not an excess of oxygen, no explosion could take place, and as a rule when an explosive was fired they did not get an excess of oxygen. He proposed to show the members a few experiments which would bear out one or two points he spoke of at a previous meeting, and there was also a new point which might be of interest to Mr. Kayll and others with regard to the delay of the ignition of gas. In the *Report*, there were cases mentioned in which delayed ignitions of gas were obtained, 1½ or 2 seconds interval occurring. Another point was the easy combustibility of coal-dust in mixtures of carbon monoxide and air. If carbon monoxide were generated by an explosive in a mine, a very dangerous factor was introduced because coal-dust would readily burn in a mixture of carbon monoxide and air. [Mr. Orsman then made experiments demonstrating how readily coal-dust could be ignited in mixtures of carbon monoxide* and air. Mr. Orsman also illustrated the three distinct phases of an explosion of mixture of coal-gas and air—the quiet burning, the vibration, and the sudden pressure or the detonation—by experiments with a long vertical glass tube. The gases were ignited at the open end of the tube, and the flame travelled steadily downwards to a certain point, where vibrations were set up, and the gases then exploded violently or were extinguished.]

Mr. A. L. STEAVENSON asked if an explosive when carefully and completely fired was liable to give off carbon monoxide, or only when it was incompletely detonated or blown-out without doing its work?

Mr. ORSMAN said it depended entirely on the explosive. Some explosives would give off carbon monoxide whether properly or badly

* The carbon monoxide used in these experiments was the product of the detonation of carbonite in a closed bomb.

fired; thus it did not matter how carbonite was fired, it would always yield about 40 per cent. of carbon monoxide.

Mr. A. L. STEAVENSON said that Mr. Winkhaus' paper was an excellent addition to the valuable information on this subject, which was being gradually stored up in the *Transactions*. He observed in the paper that all the shots were fired without stemming, but as there was a great deal of difference between shots stemmed and unstemmed, he thought the experiments should not be concluded without a series with shots properly stemmed. It was stated in the paper that "the impalpable coal-dust used in the experiments, taken from the sweepings of the screening-apparatus of a bituminous coal-pit, was throughout obtained from the same source. Its uniformity was constantly checked by means of analyses and by comparative shots fired with a given quantity of an explosive whose ignition-limit was known." It was also stated that "the physical properties of the dust (fineness of state of division, etc.) were, as a consequence of its origin, uniform throughout." It was desirable, however, to have some gauge to know what the fineness of the dust really was, for dusts varied very considerably. A useful remark was, "in the presence of gaseous mixtures containing small percentages of fire-damp (say about 2½ per cent. of marsh gas) the safety of the greater number of explosives was found to diminish in a very remarkable degree." Blasting-powder appeared to have been excluded from the experiments, on the ground that it was highly dangerous in all fiery and dusty mines, and should therefore be no longer used in such mines. No doubt they were gradually all coming to that opinion, and although many people say that all the safety explosives would give a flame and explode gas, they would not do so to the same extent as powder, which was undoubtedly the worst of all explosives. The composition of westfalit differed somewhat from the analysis of the similar explosives used by the Explosives Committee. The experiments with electric detonators were very satisfactory. He thought it had been contended in regard to some tests that the question of the size of the detonator had a good deal to do with the fact of whether ignition of the explosive was caused or not. The question of the paraffined cover was important, for if the workmen had to take off these covers it would considerably mar their effect in use.

Mr. W. C. BLACKETT said that the various discussions which they had had upon this subject served to strengthen the objection he took at first to the assumption that all the observations of light by the experimenters were due to dangerous flame. To illustrate his meaning, although

was a little far-fetched, let him take the case of a fire-fly or glow-worm. If a number of these insects had been seen in the experimental apparatus it was quite likely that the observers would have said that they had seen sparks or even small flames, although what they really had seen would be light at a temperature which could not be said to be very hot. There were other lights, reflected lights, phosphorescent lights, and so on, which were no guides to temperature, and temperature was the main thing which the Explosives Committee ought to have observed. Mr. Hedley at the last discussion said that if a detonator was fired in a darkened room, flame could always be seen, and that flame might or might not be able to ignite gas. Now, the *Report* in connexion with detonators did not, he believed, ever use the term flame, it was always the infinitely better and more accurate term "flash," so that it was an innovation now to say that a detonator fired in a darkened room could always be seen to yield flame. The temperature applied (or the so-called flame) must have time and bulk or the gas would not be inflamed. This feature seemed to suggest that the different inflammabilities of coal-gas and pit-gas would be accounted for by the difference in their specific heats, rather than by the difference between their temperatures of ignition. Nothing had been said in the discussion which altered his opinion, that the statements made by the Explosives Committee to the effect that no greater safety was obtained by extra stemming was most misleading. All that could legitimately be claimed by the Explosives Committee was that in their experiments with plastic clay they had been unable to observe any greater safety with extra lengths. The remark as to stemming then only applied to plastic clay, and should not be allowed to apply to other and better forms of stemming. However desirable plastic clay might be as a stemming on account of its incombustibility, it appeared to him to be one of the worst (with the exception, perhaps, of water) for physical reasons. He thought that they had had in the experiments shown by Mr. Orsman a good argument bearing upon stemming. It must be admitted as a fact that, if the explosive had plenty of work to do, there would be less heat. If there had been something used which would have adhered better to the sides of the cannon, it would have given the explosive more work to do, and the extra length of stemming would have reduced the heat, and thereby have afforded greater safety. The experiments of the Explosives Committee did not to his mind so strongly point to condemning extra lengths of stemming as to condemning plastic clay, and he for one would have felt obliged had they retried some of their experiments with other substances, such as seggar-clay borings, which would

have better resisted ejection owing to the greater side-friction. Mr. Hedley on page 221* appeared to refer to something which he (Mr. Blackett) had said at a former discussion, where he threw some doubt on the value of the experiments because the conditions were not those of the mine, referring to the use of the diaphragm. Mr. Hedley said that he (Mr. Blackett) admitted such conditions might exist. That was not so. All he admitted was that they might be approximately produced, but that to do so would be as careless as using an imperfect safety-lamp. Members would perhaps agree with him when he explained that the only conceivable way in which the condition of the diaphragm could be reproduced in the mine would be by carefully bratticing a gallery, thus shutting off all air access and then firing the shot. Mr. Hedley also said that the Explosives Committee did not intend to compare gunpowder with other explosives in the Trauzl blocks. Probably then it was owing to an inadvertence that four experiments with gunpowder had been included in Table I. In the same table, in experiment No. 10, with ammonite, it was a curious fact that although the hole in the leader block was said to have been driven right through, yet, what one would naturally suppose to be an indefinite cavity, had been accurately measured as 813 cubic centimetres.

Mr. M. WALTON BROWN said that he would like to draw one or two comparisons between the results given in Mr. Winkhaus' paper and those recorded in the *Report* of the Explosives Committee. The first was with regard to the length of flame which Mr. Winkhaus said did not exceed 6 or 7 inches, but much greater lengths were recorded in the *Report*. The pit-gas from the Schalke colliery could be very readily ignited, while the Hebburn pit-gas was not so readily ignited. In Mr. Winkhaus' paper, the German names of the explosives had been retained so that the difference in composition referred to by Mr. Orsman might not be overlooked. Mr. Steavenson made some remarks with reference to the difficulty of removing the outer cases of the explosives, but this was not so difficult as might at first sight appear, as two paper covers were used on such explosives as westfalit, etc. It occurred to him that perhaps Mr. Kayll or Mr. Hedley might give some further information as to the casing of the explosives used at Hebburn. A special danger arising from coal-dust being left in shot-holes was incidentally mentioned in Mr. Winkhaus' paper; it was stated that if cartridges were placed 6 or 7 inches down a borehole, ignition of the dust-laden atmosphere could be

* *Trans. Fed. Inst.*, vol. ix.

produced if a very small quantity of coal-dust were lying within the borehole in front of the explosive.

Mr. J. L. HEDLEY (H.M. Inspector of Mines, Newcastle-upon-Tyne) said that in no case were the original covers kept on the explosives in the Hebburn experiments, a brown paper cartridge being made for each explosive. Mr. Blackett raised the point as to how they obtained the measurement of the cavity produced in the Trauzl leaden block by a certain explosive. The explanation was very simple: a small hole was blown into the bottom of the leaden block, it was simply covered up, and the measurements were taken as if the hole had not been made. With reference to the flame or flash of detonators, possibly he had used the word "flame" when "flash" was intended, and he should have been careful to make the distinction. He had sufficiently explained this matter before, and he could only say what they thought they had seen, and they could not help it if other gentlemen did not think they had seen flame. With respect to the stemming, he had before explained that the Explosives Committee agreed to use a certain kind, and the results given were simply comparative; with 1 inch of stemming they got similar results, so far as safety was concerned, as with 8 inches. They did not say that they would get similar results with other sorts of stemming, but the stemming used was that which was approved of by the manufacturers of the explosives and largely used in mines throughout the country. It might or might not be satisfactory, but what they had to prove was whether the stemming recommended was safe or not, and the results were recorded in the *Report*.

The PRESIDENT said that this subject of explosives had occupied much attention lately, but they could not overrate its importance. Legislation was likely to take place very shortly with reference to the question of shot-firing in mines, and therefore the importance of learning as much in anticipation of that legislation would be patent to all, and each member of the Institute should take advantage of the great opportunities which were now afforded them in this direction.

Mr. WINKHAUS wrote that Mr. W. J. Orsman asserted in his remarks that the experiments described in his (Mr. Winkhaus's) paper were very incomplete, because it had not been determined whether every cartridge had been fully detonated. To that circumstance, he (Mr. Orsman) attributed the failure in many cases of the various explosives to bring about ignition of the pit-gas. He (Mr. Winkhaus) had frequently noticed incomplete detonation of the explosives, particularly in the case of nitrate-of-ammonium explosives, but he had always regarded such

experiments as failures, and had not recorded them in his paper. Imperfect detonation chiefly occurred when the samples of explosives, through having been long warehoused, had become damp. But whenever with the strongest detonators (No. 8), perfect detonation did not take place, the explosive in question was put aside and fresh samples for further experiments were immediately obtained from neighbouring collieries. The phenomena which accompany incomplete detonation of the explosive could hardly escape the eye of the observer. In such cases one finds on the floor of the gallery, and particularly in the bore-hole of the cannon, remnants of the explosive more or less abundant. Now, as all the shots were fired in his (Mr. Winkhaus') presence, and as, moreover, by scraping out the bore-hole and examining the floor of the gallery, he was able to ascertain unmistakably whether remnants of the explosive were to be found or not, he regarded it as quite out of the question that a failure to explode could have escaped his attention. Nor did he (Mr. Winkhaus) admit that, in the case of a single one of the experiments recorded in his paper, there could have been incomplete detonation of a cartridge. The liberation of carbon monoxide had attracted his (Mr. Winkhaus') attention when investigating the after-damp produced by the explosives. The experiments relating to their influence on the explosibility of coal-dust had been already begun, but were not yet completed. With regard to Mr. A. L. Steavenson's suggestion that it would be desirable to have some standard of the fineness of the dust used in the experiments, he (Mr. Winkhaus) would add that this was the impalpable (flying) or upper dust (an expression not rendered in the translation) from a screening-apparatus. By this he (Mr. Winkhaus) meant the dust thrown up by the shaking of the screens, at first it remained suspended in the air and subsequently was deposited in the vicinity of the screen on the floor, the timbering and such like. This was an extremely fine meal-like dust, whose physical properties should correspond exactly to those of the dust in the pit, on the sides, roof, timbering, etc. The fineness could not be measured. On shaking it through a No. 90 sieve (8,100 meshes per square inch) no residue was left on the sieve. The experiments with stemming would be commenced very shortly, as stated in his paper.

DISCUSSION ON MR. W. O. WOOD'S PAPER ON "THE MURTON COAL-WASHER."*

The PRESIDENT said that the apparatus erected by Mr. W. O. Wood was a very simple and complete coal-washer. The chief novelty was that the water travelled in the opposite direction to the coal.

Mr. W. C. BLACKETT asked if the cubical capacity of the settling-tanks and the amount of coal washed per day could be stated?

Mr. W. O. WOOD said that the four settling-tanks had practically a total capacity of 1,000 cubic feet for 400 tons of coal washed per day. The sulphur, as a rule, decreased in proportion as the amount of ash decreased. If a coke contained 7 or 8 per cent. of ash, it might contain 1·50 per cent. of sulphur; if that ash were reduced to 5 per cent., the sulphur would be reduced in similar proportion, but in the long run, however low they got the ash, there still remained a fixed quantity of sulphur chemically combined with the coal which it was impossible for any operation of washing to reduce. The ash was mentioned in the paper as being 5·39, but since then the ash had been reduced by nearly 1 per cent., and it was accompanied by a decrease of sulphur of about 0·2 per cent.

Mr. A. L. STEAVENSON said that when in Germany some years ago he saw a coal-washer which did its work excellently—about 500 tons per day——but the cost was £10,000. No doubt the Murton washer and other washers in this country would do the same work equally well for £300 or £400.

The PRESIDENT said that a large proportion of the cost often attributed to a coal-washer was due to other portions of a complete plant. The cost of the Lührig washing-plant at Evenwood colliery was often referred to as enormous, but it included a great amount of plant unconnected with the washing of the coal.

Mr. THOMAS BELL (H.M. Inspector of Mines, Durham) said that the large amount of money expended at Evenwood was not for the washer alone, but included appliances for screening, cleaning, crushing, draining, and storage of the coal afterwards.

Mr. W. C. BLACKETT said that, as a matter of fact, if the Lührig washer alone were compared with other washers it would be found to be no more expensive. It was not fair to take the cost of a large plant and compare it with the cost of a washer. He had had an experience of a washing-

* *Trans. Fed. Inst.*, vol. ix., page 42.

plant, the plan of which would have been applicable to most washers, and he sent out the plans adapted to the washers of several different manufacturers, requesting prices for their particular washers, and out of three or four tenders the Lührig was the cheapest. He might add that the Murton washer was not then invented. The tenders included the cost of the whole of the machinery connected with the washer, and designed to work with it.

Mr. W. O. Wood said that the Murton washer was designed, owing to the excessive cost of other washing-plants.

Mr. J. M. Liddell said that the principle of carrying the water in opposite direction to the coal washed was largely used in (frue-vanner) gold-washing. The chief difference was that the apparatus had in addition an oscillating motion sideways, so as to secure more perfect separation of minerals, and did not require ripple-bars.

DISCUSSION ON DR. R. BRUNCK'S PAPER ON "THE BRUNCK COKE-OVEN AND SYSTEM FOR THE RECOVERY OF BYE-PRODUCTS,"* AND MR. JOHN H. DARBY'S PAPER ON "THE SEMET-SOLVAY COKE-OVEN."†

Mr. W. C. Blackett said that since the last meeting he had had occasion to peruse some figures in connexion with a proposal to erect Semet-Solvay coke-ovens at an English colliery. Being dubious as to the facts, he took the basis figures used for the Semet-Solvay coke-ovens and applied them to a plant of coke-ovens, such as he imagined would be placed in the category of "despised beehives," and he discovered that where the Semet-Solvay would give 70 per cent. profit the beehive coke-ovens would give 60 per cent. He need not remark that these ovens had never made such a profit, but on paper they were able to do so, and this without considering the value of waste gases firing boilers.

Mr. B. Dodd (Durham) said that in the two papers under discussion much importance was attached to the thick middle walls of the ovens, which the writers state render the ovens very easy to repair; while at the same time they tell us they have had little or no experience with regard to repairs, so they are hardly in a position to speak as to the ease with which these can be made. He (Mr. Dodd) thought, from the construction

* *Trans. Fed. Inst.*, vol. ix., page 47. † *Ibid.*, vol. ix., page 54.

of the ovens, that before the middle walls could be repaired, the inside lining must be removed, and this would entail great labour and expense. In Simon-Carvès ovens, walls of moderate thickness have been found sufficient for all reasonable requirements. At Bearpark cokeyard, a 4 tons locomotive engine and a train of coal-tubs weighing 25 tons have been run directly over the tops of the ovens without doing harm to the arches or side walls of the ovens. The thick walls are said to act as heat-receivers, taking up and giving out heat as the coking proceeds. If this be so, the temperature of these walls will rise and fall, thus causing expansion and contraction of the brickwork, with serious opening of the joints of the same. In Simon-Carvès ovens, a uniform heat is maintained in the flues, as the ovens are loaded alternately, and as long as there is volatile matter to be driven off, the temperature in the ovens does not rise above the boiling-point of the distillate, the heat passed being in volume or intensity as is required and producing as little expansion and contraction of the brickwork as possible. In considering plants for the manufacture of coke, and for the extraction of bye-products, he (Mr. Dodd) might point out that the oven was only one portion of the plant, and perhaps the least important part. The cost of the plant and the speed and economy of working depended upon the power of the condensing-plant; the speed of burning and the total cost being governed by the quantity of coal carbonized more than by the number of ovens; the loss through minor stoppages and the cost of working would be greater the more rapidly the ovens are drawn. In the case of rapidly-drawn ovens, larger hoppers for the storage of coal are required, and also increased siding and waggon accommodation, and owing to the limited bench-area the coke has to be more rapidly removed, entailing increased Sunday and holiday labour, beside the increased useless expenditure of labour in stocking coals and coke, as compared with slower working ovens; in cases where the whole of a colliery's output of coals is coked, this becomes a very serious matter. In the Bearpark coke-ovens, coals have been tested from nearly all parts of the globe and have been found to vary very greatly in the coke and bye-products produced, and in the time required for carbonization when treated in an exactly similar manner—some coals coking from ten to twelve hours sooner than a similar quantity of coals from other localities. Many engineers attach too much importance to the ovens and too little to the nature of the substance put into them; looking upon coal as if it were inorganic matter instead of what it really was, a highly complex organic compound. He was of opinion that a little thought would show that in these bye-products recovery-plants, they were really treating the preserved and

fossilized vegetable growths of the Coal-measures, with the object of obtaining from them their colours and essences, and their medicinal properties. And he thought that, when they considered the vast area of country over which the great forests of the Carboniferous time extended, they would expect to find great variation in the nature of the plant life; that grown on the low lands near the sea-coast differing considerably from the vegetation of the inland and more elevated districts. The colours with which coal-tar dyes were competing, and which were being driven out of the market, were the organic colours which are produced by certain existing forms of vegetable life, such as: aloes from India, alkanet from the Mediterranean, arnatto from Mexico and Brazil, barwood from Africa, camwood from Africa, sanderswood from Africa, Brazilwood from Brazil, Chinese greens from China, ericine from Europe, lac from India, madder from Asia and Central Europe, munyeet from India, orchella from America, Canaries, and Africa, cudbear from Europe, litmus from Europe, quercitron from North America, safflower from Europe and India, woad from Europe. In obtaining these colours from the vegetation of the present time, they extract them from the plants which yield them the most readily and cheaply. No doubt all plants contain colouring-matter, but one would not expect to extract colours from vegetable matter taken indiscriminately, with commercial success, and for this reason they are imported from foreign countries. In dealing with the plant life of a bygone age the same discretion should be used, and colours extracted from those coals which contain the fossilized remains of the plants which produced the colour molecules. The same difference of organic growth which gave rise to the variation in the colours of the plants will affect the nature of their carbonized remains, and produce different cokes in like manner as the different woods produce different charcoals, such as that from box and willow. With regard to the value of retort coke in the blast-furnace, he (Mr. Dodd) was not a blast-furnace manager, but he had attended several meetings of the Iron and Steel Institute, and carefully studied their *Transactions*. He thought that any one who had done so would be inclined to allow that in Cleveland there were several of the most eminent and scientific ironmasters in the world, and that they were also keen business men who knew the value of what they used, Yorkshiremen being amongst Englishmen proverbial for their business capabilities. Well, during the last ten years, these ironmasters, who hold a foremost position amongst metallurgists and business men, had used in their furnaces 77 per cent. of coke made in Simon-Carvès ovens, and had paid prices as high

as those paid for beehive coke during the same period, and no doubt had found it to their advantage to do so. Although the coke made in the Bearpark Simon-Carvès ovens had given satisfactory results, it did not follow that equal success would attend all plants of that description. There are instances where large amounts have been expended over similar plants, which have not proved altogether successful, chiefly through the nature of the coal used.

Mr. A. L. STEAVENSON said he would like to ask if the Simon-Carvès coke could be sold for blast-furnace purposes at the higher price and as freely as common coke?

Mr. DODD said that was so.

Mr. A. L. STEAVENSON—And did the yield of coke correspond to anything like 75 or 80 per cent. of the coal?

Mr. DODD—About 78 per cent.

Mr. W. C. BLACKETT said that while retort coke was perhaps known to be dense, yet density was not all they had to deal with, and what was wanted was not closeness of the cells to one another, which formed density, but hardness of the walls of the cells to bear the burden. In blast furnaces where retort coke was being used, were they as high throughout as the highest furnace using beehive coke?

Mr. DODD said the retort coke would carry any amount of weight; if anything it was perhaps too dense, and gave better results in smaller pieces. It had been used in both high and low furnaces.

The discussion was then closed.

DISCUSSION UPON MR. B. H. THWAITES' PAPER ON "ENGINEERING HELIOGRAPHY."*

Mr. B. H. THWAITES wrote that the developing bath consists of a flat vessel slightly larger than the print, and some 2½ inches deep, made of wood, and treated with a varnish applied to a thickness of ⅛ inch—the varnish being composed of ½ lb. of brown resin and 2 ounces of beeswax. The Thwaite developing bath (Fig. 10, Plate III.) is of a vertical form, so that when the prints are immersed the redundant and unacted on iron salts are deposited at the bottom. For acid solutions, the bath should be lined with vulcanite, enamelled iron, or be made of earthenware.

The meeting then terminated.

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* *Trans. Fed. Inst.*, vol. ix., page 69.

MIDLAND INSTITUTE OF MINING, CIVIL, AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD AT THE ROYAL VICTORIA STATION HOTEL, SHEFFIELD, MAY 18TH, 1895.

MR. JOHN NEVIN, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and signed.

Mr. A. M. CHAMBERS (Thorncliffe Collieries) moved that a vote of condolence and sympathy be sent to Mrs. Mitchell and the members of the family in the very great loss which they have sustained. Mr. Joseph Mitchell had rendered immense service to that Institute. He was Secretary and Treasurer for a great number of years, and though the Institute seemed to have dwindled away to almost nothing at one time, it was due to his exertions that the Institute had been brought round to the position which it occupied that day. Mr. Mitchell was a thoroughly representative Yorkshireman, straightforward, somewhat blunt, perhaps, always true to his own convictions, and ready to support those who were equally straightforward.

Mr. M. HALL (Lofthouse Colliery) seconded the motion, and said that he had known Mr. Mitchell for the last seventeen or eighteen years, and felt that the profession had lost an able mining engineer. He had met Mr. Mitchell frequently in arbitrations and other matters, and found him as capable in these as he was affable in social life.

The PRESIDENT, in putting the motion, said that he had known Mr. Mitchell for the last twenty years, and could quite bear out what had been said. As a younger member of the Institute he had received great kindness from Mr. Mitchell, and should always remember him.

The motion was carried unanimously and in silence.

The following gentlemen were elected, having been previously nominated :—

MEMBERS—

Mr. WILLIAM HAIGH, Colliery Owner, Dewsbury.
Mr. H. G. SOAR, Colliery Manager, Frystone Collieries, Castleford.
Mr. HARRY ERNEST HUGHES, Mining Engineer, 9, Willow Terrace Road, Leeds.
Mr. RANKIN KENNEDY, Electrical Engineer, Phœnix Dynamo Manufacturing Co., Hubert Street, Leeds Road, Bradford.
Mr. G. F. FINDLAY, Mining Engineer, Broughton and Plas Power Collieries, Wrexham.

ASSOCIATE MEMBERS—

Mr. HERBERT INGOLD, Manager, Wire Rope Works, Tinsley, near Sheffield.
Mr. JOHN EDWARD DAVENPORT, Iron Merchant, The Elms, Oulton, near Leeds.

STUDENT—

Mr. JOSEPH E. MITCHELL, Mining Offices, Barnsley.

Mr. W. WASHINGTON read the following paper on "An Outburst of Gas at Mitchell Main Colliery" :—

AN OUTBURST OF GAS AT MITCHELL MAIN COLLIERY.

By W. WASHINGTON.

Outbursts of gas have on many occasions been brought to the notice of the members of this Institute, and the author in presenting the following particulars has only one object in view, *i.e.*, of giving another illustration of the great difficulty of foreseeing accidents in mines.

On April 24th, 1895, an outburst of gas took place at about 5·30 p.m., in the north-west district at the point marked A (Fig. 1, Plate XVII.). It appeared that the afternoon deputy, on making his usual inspection of this district about 3·30 p.m., found two working-places on the weight at the points marked B and C, and, having withdrawn the workmen from these places, fenced them off. On completing his inspection of the district, the deputy was informed that an outburst of gas had taken place. He proceeded to the spot, and found that on the intake side of the outburst the working-places were fouled with gas for a distance of 120 feet along the face at the point marked D, and for a distance of about 360 feet from the face down the main hauling gate-road at the point marked E. He then cautiously proceeded to examine, as far as practicable, the return side and found that all the working-places were full of gas; the return airway being fouled to the upcast pit, a distance of 7,200 feet. On the intake side, at and near the point F, two safety-lamps were extinguished; and on the working-face on the return side from the point C to the end of the face, two lamps were extinguished; and from the end of the face, a distance of 1,800 feet on the return side, five lamps were put out.

The writer on hearing of the outburst, about 7·30 p.m., descended the pit and was informed that gas had shown itself in the lamps near the upcast pit-bottom: on examination, however, no gas was detected at this point. He then, in company with two deputies, travelled the main return-airway towards the place of the outburst, and observed a blue cap on his safety-lamp at a distance of 4,200 feet from the upcast pit-bottom. He then went by way of the intake-airway to the intake side of the outburst, and at the point G found gas in the new cross gate-road. The ventilation was then largely increased, but the gas continued to be given off very strongly.

A competent man was put in charge of each shift, and the feeder of gas gradually grew weaker, until on Saturday, May 4th, the workings on the return side were reported clear of gas, which had nearly spent itself at the end of ten days, and the workmen were re-admitted.

At the time of the outburst, from 18,000 to 20,000 cubic feet of air per minute were passing through this district, and at the actual place of the outburst from 14,000 to 16,000 cubic feet of air per minute were passing, the quantity at the upcast pit-bottom being upwards of 120,000 cubic feet per minute.

The greater portion of the gas appeared to come from the floor, and for a distance of 150 to 200 feet along the face, the floor was considerably lifted, the roof had also subsided, thus reducing the ordinary height of the road from 5 feet 6 inches to about 3 feet. The blower still continues to give off a small quantity of gas, but this is being successfully coped with.

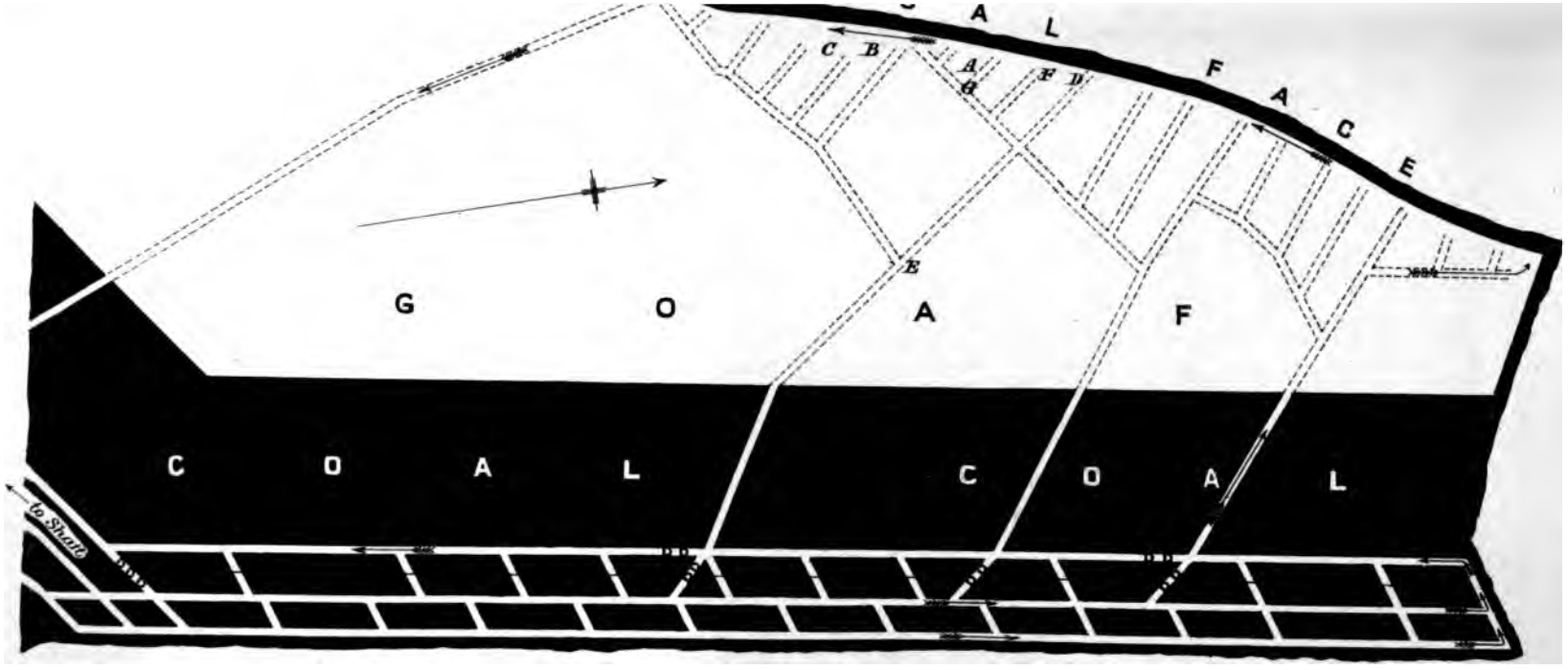
The deputy said that "the concussion appeared to shake the separation-doors at a distance of 840 feet from the point of the outburst."

The system of working is longwall, the working-line being half-end and half-bord. The gate-road packs are 9 feet wide on each side, and the gate-roads are placed 66 feet apart, with cross gate-roads cutting off the gate-roads about every 250 feet. The working-face, at the place of the outburst, was about 540 feet into the goaf, and its length about 1,200 feet.

Protector Mueseler lamps are in use throughout the colliery. It may also be recorded that no gas had been previously reported in the immediate locality of this outburst.

Mr. WASHINGTON, in reply to questions, said that the roads on the face rose about 1 in 36 to the north-west. The deputy, who was sitting some distance away, said that the doors slightly opened owing to the concussion at the time when the outburst happened. There was a fall of roof at the point A.

Mr. A. M. CHAMBERS (Thorncliffe Collieries) moved a vote of thanks to Mr. Washington for his paper. The occurrence described was very similar to many other outbursts of gas. The last outburst at the collieries under his (Mr. Chambers') charge, caused a small explosion by which two workmen were killed. At that time (a good many years ago) Stephenson safety-lamps were in use. One satisfactory matter in



Scale-264 Feet to 1 Inch

Midland Institute of Mining, Civil & Mechanical Engineers. Transactions 1894 5.

And.w Reid & Comp.y Ltd. Newcastle on Tyne.

Vol. XIV. Plate IV.

connexion with recent outbursts of gas was the fact that they proved that the type of safety-lamp generally used in South Yorkshire collieries was practically safe under such circumstances. The Mueseler and Marsaut safety-lamps were safe under such conditions which came upon the workmen so suddenly, conditions which might prove very disastrous if the types of safety-lamps formerly used were still employed.

Mr. A. A. ATKINSON (Barrow Collieries) seconded the motion.

Mr. WORDSWORTH (West Riding Collieries) asked if there had been a previous weight between the fast side and the working-face, or whether this was the first occurrence?

Mr. WASHINGTON said there had been several heavy weights of the roof before this one.

Mr. A. M. CHAMBERS asked if there had been a thorough ground weight?

Mr. WASHINGTON said that this was the chief ground weight which had occurred in that particular portion of the face.

Mr. OVEREND (Barnsley) asked whether after the places B and C were on the weight, a continual weight took place until it reached the point A, or whether a fresh weight took place at A?

Mr. WASHINGTON said B and C were on the weight, and the weight spread more to the north than towards A, but it ultimately affected the point A.

The PRESIDENT remarked that in several cases of which they had records, as at Strafford main collieries, outbursts took place when the workings had only proceeded a certain distance, or when the area of goaf did not exceed 8 or 10 acres?

Mr. W. HOOLE CHAMBERS (Tankersley Colliery) asked whether the gas continued to come from the floor or the roof?

Mr. WASHINGTON said that the gas was given off from both floor and roof. A deputy's lamp was extinguished when on the floor. A stick could be pushed into the crack in the floor.

Mr. THIRKELL (Oaks Collieries) asked if there was an absolute fall at the point B and C, or was it only a weight?

Mr. WASHINGTON said that the roof did not break at those points, but previously there had been a very heavy fall.

The motion was agreed to.

Mr. E. W. THIRKELL read the following notes on "Watering Underground Roadways," etc.:—

WATERING UNDERGROUND ROADWAYS, AND A LIGHTED LUCIFER MATCH AT THE NEW OAKS COLLIERIES.

By E. W. THIRKELL.

Watering Underground Roadways.—The watering of the roadways at the New Oaks collieries, is effected in the following manner:—A length of delivery-pipes in connexion with a pump is laid along the engine-plane. From these pipes a supply of water is taken by two tubes (1 inch in diameter) to which nozzles are attached. The water issues from the nozzles in the form of a fine spray; this travels in the air for about 120 feet and damps the dust on the sides, roof, and floor for that distance. Three of these nozzles are placed at intervals of about 1,200 feet.

Another tube (1 inch in diameter) is taken from the delivery-pipes to a drift, and is there fixed to two perforated pipes similar to those used in water-carts. The perforated pipes are used to dampen the coal on the full corves, as they come out of the landings from the workings. This damping of the coal prevents the dust from flying about in the engine-plane, and as the intake air in the drift travels at a speed of about 9 miles per hour, and the load of corves travels at a speed of about 12 miles an hour, it will at once be recognized that a considerable current is produced. If a load of corves travels unwatered it is at once detected.

The levels are watered by means of the Fisher water-tub, in which brushes are fixed behind, and throw the water all over the place when it is drawn by a horse. One tub will hold about 110 gallons of water, and will dampen a length of about 1,500 feet of roadway. It is, however, intended to put in tubes and use the nozzles in these levels.

Advantage is taken of the presence of water in any places in the pit, and it is used to wet the sides, etc., so as to thoroughly dampen the dust. The dust is carefully cleared away for a length of about 60 feet below each nozzle, and this distance is kept constantly wet.

Lucifer-match Incident.—On June 9th, 1892, two workmen (datallers) were enlarging a return-airway, by removing the floor, which had lifted. Whilst shovelling the loose material into a corf, one of them threw a piece of stone with the small stuff and directly afterwards saw a light in

the corf, and examination proved that it came from a lighted match. The match had been thrown in with the rubbish, and became ignited by one stone striking it against another piece. The match had probably fallen out of a workman's pocket. An endeavour was made to find out how it came to be where it was, but without success. This occurrence shows how it is possible for accidents to occur in the most unforeseen manner.

Mr. A. A. ATKINSON (Barrow Collieries) asked whether three nozzles were sufficient to thoroughly water the road for a distance of 1,200 feet, and what was the pressure of the water issuing from the nozzles ?

Mr. THIRKELL said the water-spray travelled for a distance of about 120 feet from each nozzle. No attempt was made to dampen the whole distance at first; it would not be desirable, on account of the floor and the roof, to put too much water in any one place, and recent investigation had shown that if there were lengths of roadways at intervals where the dust was entirely removed and which were well watered, then these places would, if they did not absolutely stop the force of the explosion, unquestionably check it. The water was used at a pressure of about 270 pounds per square inch.

Mr. A. M. CHAMBERS asked whether the water had any detrimental effect on the floor or roof ?

Mr. THIRKELL said that the spray was so very fine that it had not had the effect of doing much damage.

Mr. J. E. CHAMBERS moved a vote of thanks to Mr. Thirkell for his notes.

Mr. HALL seconded the motion, which was agreed to.

Mr. M. HALL read the following report on "Experiments with Explosives":—

EXPERIMENTS WITH EXPLOSIVES.

BY E. W. THIRKELL, A. A. ATKINSON, AND MATT. HALL.

The following tables contain the results of experiments for safety and strength with explosives for use in coal-mines, at the experimental station at Messrs. Pearson & Knowles' Coal and Iron Company, Limited, Ince Hall collieries, near Wigan, Lancashire :—

TABLE I.—SAFETY TESTS.

Date.	Shot No.	Name of Explosive.	Weight of Explosive used.	Coal-dust from what Mine.	Quantity of Coal-dust used.	Number of Detonator.	Results.
1895.			Ozs.		Cubic Foot.		
Feb. 22	1	Ammonite ...	12	Arley	½	6½	No explosion. No flame.
,, ,,	2	Carbonite	12	Arley	½	6½	No explosion. No flame. No ignition. Some say there were sparks, but we saw none.
,, ,,	3	Roburite	12	Arley	½	6½	No explosion. No flame. Some say there were sparks, but we saw none.
,, ,,	4	Roburite	12	Arley	½	6½	Violent explosion, ignition, and large flame.
,, ,,	5	Bellite	12	Arley	½	6½	Violent explosion, ignition, and large flame.
,, ,,	6	Westphalite ...	12	Arley	½	6½	No explosion. No ignition. (Incomplete test.) Boiler cleaned out before firing No. 6 [Westphalite].
,, ,,	7	Westphalite ...	12	Arley	½	8	No explosion. No ignition. No flame. Sparks seen.
,, ,,	8	Tonite	12	Arley	½	6½	Violent explosion, ignition, and large flame.
,, ,,	9	Ammonite ...	12	Arley	½	6½	Violent explosion, ignition, and large flame.
,, ,,	10	Carbonite	12	Arley	½	6½	No explosion. No ignition. No flame.
,, ,,	11	Ardeer powder ...	12	Arley	½	6½	No explosion. No ignition. No flame.
,, ,,	12	Compressed powder	8	Arley	½	6½	Violent explosion, ignition, large flame.
,, ,,	13	Dynamite	2	Arley	½	6½	No explosion. No ignition. Sparks.
,, ,,	14	Westphalite ...	12	Arley	½	8	No explosion. No ignition. No flame. Sparks
,, ,,	15	Westphalite ...	12	Arley	½*	8	No explosion. No ignition. No flame. Interval of 15 minutes between Nos. 14 and 15

* Plus 9 per cent. of gas.

TABLE II.—STRENGTH TESTS.

35 lbs. Projectile Fired from a Cast-steel Mortar set at an Angle of 55 degs.			
Shot No.	Name of Explosive.	Weight of Explosive Charge.	Result. Distance the Projectile was thrown.
		Grammes.	Feet.
1	Ammonite	10	424½
2	Carbonite	10	241½
3	Roburite	10	492
4	Tonite	10	249
5	Westphalite ...	10	492
6	Bellite	10	516
7	Blasting-powder ...	10	165
8	Ardeer powder ...	10	84
9	Dynamite	10	498

No. 6½ detonators were used in these experiments.

Safety-tests.—In all cases the explosive was fired into a boiler in a horizontal position, 30 feet long and 5 feet 10 inches in diameter, open at one end, and fitted with three windows A. A gas chamber B was formed by means of an air-tight partition placed at a distance of 8½ feet from the closed end of the boiler.

At the closed end, near the bottom of the boiler, a cast-steel mortar C (the bore of which was 2 feet long and about 1½ inches in diameter) was fixed in mason-work. (Figs. 1, 2, and 3, Plate XVIII.)

All the charges were placed in the mortar, and were unstemmed, which is the most favourable condition for the explosive to show flame or ignite coal-dust. We did not witness the handling of the charges.

The shots were fired by electricity.

Observations were taken at a considerable distance from the boiler.

Only one experiment (No. 15) was tried with a mixture of coal-gas and air, a portion of the boiler, 225 cubic feet, being screened off by several thicknesses of brattice cloth. In this portion of the boiler there were also two openings at the top also covered by brattice cloth, and 9 per cent. of gas measured by a gas-meter was introduced.

There was no mechanical means used to diffuse the gas with the air.

The coal-dust was put into a box, ½ cubic foot being used each time, and carried into the boiler by a workman, who threw it out near the mortar end in such a way as to leave as much as possible in suspension when the charge was fired. No mechanical means to agitate the dust were used.

The dust in some of the earlier experiments was not perfectly dry, owing to its having been exposed to the atmosphere for several hours. The temperature was about 34 degs. Fahr.

In No. 6 experiment with westphalite, a No. 6 detonator was used, the result being no explosion and no ignition. This, however, was an incomplete test, as only one cartridge was fired.

In experiments Nos. 7, 14, and 15, with westphalite, No. 8 detonators were used, whereas in all the other experiments No. 6½ detonators were used. Sparks were seen in Nos. 7 and 14, but not in No. 15 experiment.

Strength Tests.—The projectile, 35 lbs. weight, was fired from a cast-steel mortar (Figs. 4, 5, and 6, Plate XVIII.) placed at an angle of 55 degs., by means of ordinary white fuze, the resultant distances being measured from the mortar to the point where the projectile first dropped. The results are shown in Fig. 7 (Plate XVIII.)

In all cases, 10 grammes (about ⅓ ounce) of the explosive was used, together with a No. 6½ detonator.

For a thoroughly reliable test of the explosives, we are of opinion that independent parties should make the tests.

Mr. M. HALL (Lofthouse Colliery) said that the members of the Institute had received an invitation to see experiments with various explosives made at Messrs. Pearson & Knowles' Ince Hall collieries, at the invitation of the Westphalite Syndicate, Limited. When the members of the Midland Institute arrived, they were asked to appoint a committee to check the results of the experiments. They felt that they were not appointed by the Institute, but on being pressed, Mr. Thirkell, Mr. Atkinson, and himself undertook the responsibility. Although they saw the experiments, they were not satisfied with the manner in which they were conducted. They were unanimously of opinion that such experiments should be made by independent engineers, that the explosives should be provided by the persons appointed to make the tests, and not by the proprietor of one of the explosives tested.

Capt. ENSOR (London) said that he had not the advantage of being present at the Wigan experiments, and even if he had been, he could not claim to have sufficient expert knowledge to criticise in any way this paper. There was, however, one part of the paper in which the writers express their opinion that trials of this nature should be perfectly open and conducted by independent persons. The Westphalite Syndicate

FIG. 8.

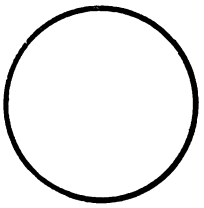
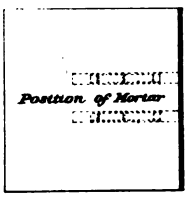
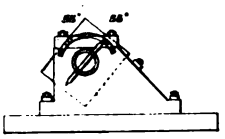


FIG. 4.

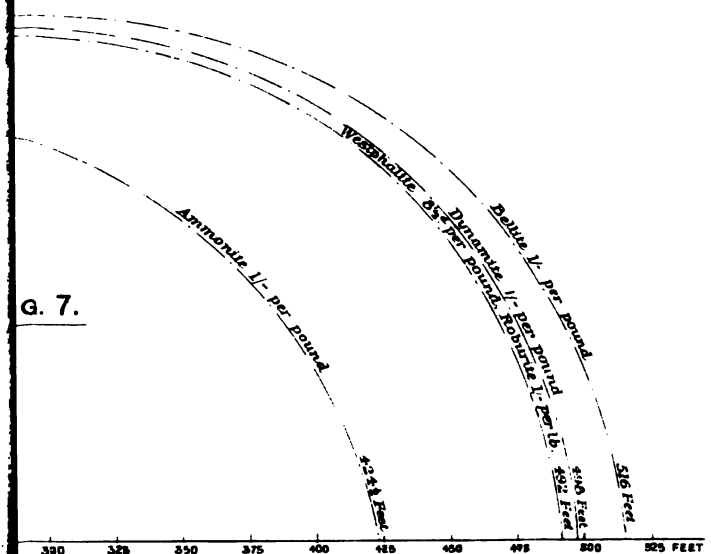


PLAN.

FIG. 5.



SIDE ELEVATION.



G. 7.

were anxious that the trials should be so, and were willing to afford every facility for satisfying any committee in this way. He (Capt. Ensor) might add that all the shots of those experiments were laid and fired by Mr. J. Elce, a member of the Institute, and not by any person connected with the Westphalite Syndicate.

Mr. OVEREND said that there was a great difference in the results of tests No. 1 and No. 9 with ammonite; possibly the difference might be accounted for by the dryness of the coal-dust.

Mr. M. HALL said that the dust used in the first experiment was not satisfactory, having been exposed to the atmosphere for some hours. Consequently a fresh sample of dust was obtained, and used in the subsequent experiments.

Mr. A. M. CHAMBERS moved a vote of thanks to the committee for their report. He agreed with the criticisms of the committee that it was desirable that experiments of this kind should be conducted by a committee of impartial mining engineers, who would ascertain on a scientific basis which was the best and safest explosive. The Council of the Mining Association of Great Britain had under consideration the very important Mines Bill introduced by the Government for dealing with the question of explosives and with the report of the Royal Commission on Explosives and Coal-dust in Mines. There was not at present sufficient information to enable the members to say that any explosive hitherto introduced was absolutely safe in a dry and dusty mine. An absolutely flameless explosive was wanted, and if any one could discover an explosive that was absolutely flameless under all circumstances, he would confer a great benefit on the mining industry of this and every other country. No such explosive had yet been discovered, and there was consequently a strong demand on the part of the miners' representatives that in dry and dusty mines no explosive should be used under any circumstances. The miners' representatives further said that it was quite as wrong to have one man in the pit as a hundred. The twelfth General Rule allowed ten men to be in the pit, but the men's representatives held strongly that if there was danger of explosion it was quite as wrong to have ten men exposed to a fatal disaster, as to have any other number, and they therefore pressed strongly that no explosive should be used in a dry and dusty mine. It would be of a great benefit to coal-owners and mining engineers if a committee of the members of The Federated Institution of Mining Engineers could be formed to conduct experiments on explosives on an independent and scientific basis.

Mr. WASHINGTON seconded the proposition, which was agreed to.

Mr. HALL, replying on behalf of the committee, said that every county ought to have its own experimental explosives station, so that each district, with its various coal-dusts, could make its own experiments.

The PRESIDENT said that the suggestion would be discussed by the Council.

The discussion on Mr. H. Wormald's paper on "Designs for Pump Valves" was closed.

The meeting was then closed.

THE FEDERATED INSTITUTION OF MINING ENGINEERS.

GENERAL MEETING,
HELD IN THE ROOMS OF THE SOCIETY OF ARTS, 20, JOHN STREET, ADELPHI, LONDON, MAY 30TH, 1895.

MR. W. N. ATKINSON, PRESIDENT, IN THE CHAIR.

The PRESIDENT read the following address :—

PRESIDENTIAL ADDRESS.

BY MR. W. N. ATKINSON.

Whilst I am desirous of conforming to the obligation imposed by custom on the President of The Federated Institution of Mining Engineers, by delivering a presidential address, I feel how little qualified I am for the task.

I wish first to thank you for the great honour that you conferred on me in selecting me to be your President, which I need not say is an honour I appreciate very highly. My thanks are also due to the members of Council and our able Secretary for the kind way in which they have advised and supported me.

I purpose devoting this address chiefly to the subject of safety in mining, and I do not think it necessary to apologize for this selection, because the promotion of safety is one of the chief objects of the various mining institutes forming The Federated Institution of Mining Engineers, and it is the phase of mining with which I am more directly concerned and interested as one of H.M. Inspectors of Mines.

The miner's calling is naturally one of the most dangerous of all industrial occupations, and the career of the mining engineer may be said to be a continual struggle with the forces of nature, to wrest from the earth her concealed treasures.

We may, however, congratulate ourselves that, relatively, the miner's occupation is now much safer than it was during the earlier part of the period for which statistics are available. Thus, according to the official statistics relating to mines under the various Coal Mines Regulation Acts (which are the statistics always referred to) the ratio of persons employed in

and about our coal mines to each death during the five years 1851-55, was 233 (death-rate of 4·294 per 1,000), whilst in the five years 1890-94 the corresponding ratio was 624 (death-rate of 1·602 per 1,000), so that during the latter quinquennium more than two and a half times as many persons were employed per life lost as in the earlier period. It may be that the foregoing comparison is not absolutely accurate, by reason of the fact that the number of persons employed during the earlier period was an estimate only, and not based on actual returns, such as have been made since 1873. It is not probable, however, that this difference would materially affect the comparison.

There are many reasons for hoping that the death-rate may be still further reduced, and a brief examination of the mortality from the different classes of accidents may enable us to see where the reduction has taken place in the past, and guide us to some extent in the endeavour further to reduce the loss of life.

The different classes of accidents will be referred to in the order adopted in the official statistics. The actual and relative importance of each class is seen by comparing the total figures under the various heads for the forty-four years embraced by the statistics.

Class of Accidents.	No. of Fatal Accidents (1851-94).	No. of Deaths (1851-94).	Percentage proportion of Deaths.
Explosions of fire-damp and coal-dust	2,126	9,706	21
Falls of roof and sides	18,226	18,814	40½
Shafts	5,505	6,263	13½
Miscellaneous, underground	7,199	8,160	17½
Surface	3,362	3,484	7½
Totals	36,418	46,427	100

Explosions of Fire-damp and Coal-dust.—Explosions of fire-damp and coal-dust account for 21 per cent of all the deaths from accidents in and about the mines under the various Coal Mines Regulation Acts during the forty-four years 1851-94. The annual loss of life from explosions varies more widely than that from any other class of accidents in mines. During the past forty-four years it has varied from 651 in 1866 to 49 in 1888, the average annual number of deaths from this cause being 220.

For several years past explosions in coal-mines have been the subject of much controversy and special investigation, owing to the advancement of the opinion that coal-dust is the chief source of danger in extensive explosions. This contention is now widely admitted, although there are

still differences of opinion concerning the relative influence of fire-damp and coal-dust in mine explosions, and as to the conditions under which coal-dust may cause or extend explosions. The subject has recently been investigated and reported on by a Royal Commission, who in their report recommend the adoption of additional precautions to prevent the initiation of explosions and their extension by coal-dust. There are strong reasons for hoping, as the conviction of the danger arising from coal-dust spreads and deepens, and when such further precautions are taken as may be found practicable, that the great loss of life caused by devastating mine explosions will be materially reduced. On the other hand, we must remember that the average depth of coal-mines is increasing, and this depth is accompanied by increased dryness, so that in the future a greater proportion of mines will be subject to danger from coal-dust than in the past.

Statistics indicate that since the time when the influence of coal-dust in colliery explosions was seriously considered, there has been a marked reduction in the loss of life from this cause.

Period.	Average No. of Fatal Explosions per Annum.	Average No. of Lives Lost per Annum.	Ratio of Persons Employed Above and Below Ground to each Death by Explosions.	Death-rate per 1,000 Persons Employed Above and Below Ground.
Ten years, 1851-60	82·0	244·1	1,008	0·992
,, ,, 1861-70	56·5	226·7	1,408	0·710
,, ,, 1871-80	42·4	268·6	1,795	0·557
,, ,, 1881-90	24·5	166·1	3,199	0·313
Four years, 1891-94	18·0	165·2	4,208	0·237

These figures show that during the whole period of forty-four years there has been a great and constant diminution in the number of fatal explosions, this diminution being one of the most striking features in the statistics of mining accidents. Previous to 1881, however, the reduction in the number of fatal explosions was not accompanied by any reduction in the number of lives lost; most of the explosions which were prevented being explosions of fire-damp alone, such as usually caused the loss of one or two lives only. The prevention of these explosions was chiefly due to the substitution of safety-lamps for naked lights. The figures also show that the ratio of persons employed to each death improved steadily up to 1880, but after that year a much more rapid improvement occurred.

To what extent the reduced loss of life since 1880 may be attributed to the recognition of the influence of coal-dust, together with the legislative requirements of the Act of 1887, which was the first to mention coal-

dust, can only be conjectured. No doubt the introduction of explosives safer than gunpowder, and the extended use of safety-lamps have contributed to this happy result, but it seems reasonable to suppose that the more accurate diagnosis of the cause of wide-spreading explosions has conduced materially to the same end.

Whether wide-spreading colliery explosions can be altogether prevented depends almost entirely on the practicability of keeping the roads in coal-mines free from dangerous accumulations of coal-dust. To what extent and in what manner this can best be accomplished remains to be proved. That there are serious difficulties in the way cannot be denied, but should these difficulties in any case be insuperable, even then it may be possible to limit the extent of explosions by dividing the mine into what may be termed "fire-tight compartments," by blocking up unused roads and clearing sections of roads in use from accumulations of inflammable dust, so as to provide dustless space-intervals too long to be traversed by a dust explosion. Our experience in the use of water for damping coal-dust is as yet too limited to enable an accurate judgment to be formed of the extent to which it is practically applicable.

Some valuable aids for the prevention of explosions in mines have been introduced in recent years. Foremost among these are the various "safety" explosives; for although it has been proved experimentally that under certain conditions these explosives are capable of igniting gas and coal-dust, it is nevertheless conclusively established that when used with proper care they are very much safer than gunpowder. The experiments which have been made (and are still in progress) by the North of England Institute of Mining and Mechanical Engineers, to test the relative safety of a number of the new mining explosives will materially aid us in comprehending the conditions requisite for their safe use. I would suggest that, when the Committee complete their examination of these explosives, they might endeavour to arrive at some conclusion respecting the amount of uninflammable material required to be mixed with coal-dust—in order to render it incapable of transmitting flame for long distances. We also require data respecting the length of the intervals free from inflammable dust requisite to arrest the extension of a coal-dust explosion in mine roadways, but this will require experiments on a much larger scale than can be made with any apparatus erected in this country. In certain German collieries it has been laid down that 164 feet (50 metres) may be considered sufficient for this purpose, from what data I am not aware, and in the absence of data it would appear possible that a greater distance might be requisite in some cases. The minimum distance required to arrest a coal-

dust explosion would probably vary with the sectional area of the road and the quality of the dust.

Improved safety-lamps have now been universally adopted, but in this respect there is room for further progress. For some time we have appeared to be on the eve of obtaining a satisfactory electric mining lamp, but as yet it is not forthcoming. With respect to the lamps in use, two points strike me as specially requiring attention. The first is the abolition of the old screw-lock and the adoption of a lock incapable of being opened surreptitiously without detection, and this is an improvement of easy attainment at small cost, applicable to most of the lamps now in use. The other point is the risk attending the use of lamps with close shields, that lamps may be taken into the mine without the gauze. That this is a real risk is proved by its occurrence in the experience of a number of persons. The remedy is not so obvious as the improved lock, and it involves some alteration in the construction of the lamps, but it is a danger deserving of serious attention. The introduction of improved gas-detecting lamps, capable of indicating the presence of very small proportions of inflammable gas, enables a more accurate estimate to be formed of the production and presence of fire-damp in mines. For some time I have used Dr. Clowes' hydrogen lamp for this purpose with much satisfaction.

Falls of Roof and Sides.—Falls of roof and sides in mines have always been, and probably always will be, accountable for more accidents and deaths than are due to any other single cause. The reason is not far to seek. It is a danger common to all mines, and one to which every person engaged underground is to some extent continually exposed. About 40½ per cent. of all the deaths from accidents in mines under the Coal Mines Regulation Acts since 1851 were caused by falls of roof and sides. The statistics show, nevertheless, that there has been a steady decline in the death-rate from falls. For the period since 1873 (the statistics being based entirely on statutory returns, it is possible to separate the number of persons employed underground from those employed on the surface) the following figures apply:—

Period.	Average No. of Fatal Accidents by Falls per Annum.	Average No. of Lives Lost by Falls per Annum.	Ratio of Persons Employed Underground to each Death by Falls.	Death-rate per 1,000 Persons Employed Underground
Ten years, 1873-82	439·4	453·4	889	1·124
,, ,, 1883-92	448·4	460·2	1,002	0·998
Two years, 1893-94	417·5	428·0	1,308	0·764

The methods of preventing accidents from falls are well known to all engaged in mining, and there is no prospect of the discovery of any new principle for the avoidance of these accidents. The only hope of improvement lies in the better application of the old methods.

The substitution of iron or steel supports for timber is likely to extend for economical reasons and may to some extent add to safety. Two outstanding questions relating to accidents by falls are (1) as to whether the timbering at the working-faces, where the great bulk of the accidents happen, should be done by the miners or by persons specially appointed for the purpose, and (2) whether timber or other supports should be applied at stated intervals in the absence of apparent danger. In Durham and Northumberland much of the timbering at the working-faces is done by officials appointed for the purpose, but in all other districts of the United Kindom this timbering is done by the miners. Durham and Northumberland occupy a favourable position with regard to the mortality from falls, but this may to some extent be due to the natural conditions being safer; and the moderate death-rate from falls in Durham and Northumberland is not so marked or exceptional as to warrant a decided opinion that it is due to the particular system, which, nevertheless, possesses great advantages apart from the question of timbering. On the other hand, there appears to be a growing feeling that by the adoption of what is called systematic timbering, greater safety would be secured, and my own experience leads me to conclude, that, conjointly with efficient supervision, this is the most likely direction in which to look for further improvement.

Accidents in Shafts.—Shaft-accidents account for 13½ per cent. of the deaths recorded since 1851. The statistics show a more marked decrease in the number of deaths by shaft-accidents than from any other single cause. Although it cannot be shown by the figures it is probable that the decrease is due in a greater degree to lessened loss of life amongst persons simply using the shafts to pass to and from their work, than to increased safety to those engaged in sinking and repairing shafts and other work connected with shafts. As in the working of railways, it is the mere passenger who runs the least risk.

The reduced number of shaft-accidents is no doubt due to the great improvements which have been effected in the fittings of shafts, the almost universal use of guided cages, automatic fences, and other improved appliances connected with the shafts and winding-engines. In many cases a very high degree of perfection has been attained in this respect. Point-

to which attention should be directed in order to prevent loss of life in shafts are the maintenance of the sides of the shafts and the shaft-fittings in thoroughly good repair; also the avoidance as far as possible of the use of hangings-on or hooking-places in mid-shaft, and where that is unavoidable the adoption of special appliances for securing safety at such places. Then the use of good ropes of adequate strength vigilantly examined, oiled and cared for; periodical recapping, and, where there is the probability of internal corrosion, extra precaution with regard to the length of time during which the rope is used—as ropes, outwardly perfectly sound, may break suddenly from this cause. The use of detaching-hooks may now be recommended with confidence, and they should be supplemented by the provision of keps or catches to arrest the fall of the cage in case the chains are broken, as may occur by the fall of a cage carried up by its momentum after the rope is detached. Appliances for arresting the fall of the cage in case of the rope breaking have not arrived at such perfection as to warrant the recommendation of their general use.

Miscellaneous Underground Accidents.—Accidents under this heading caused 17½ per cent. of the deaths recorded since 1851, and the death-rate shows a tendency to increase rather than the contrary. They are usually single fatalities, caused by the use of explosives, suffocation by gases, accidents in connexion with the haulage of minerals, and arising from sundry other miscellaneous causes; but occasionally a large loss of life occurs from fires underground and by sudden irruptions of water, both of which are included with miscellaneous underground accidents. A large proportion of these accidents occur in connexion with the movement of tubs, and in many cases both safety and economy would be increased by the adoption of improved underground rolling stock and more perfect haulage roads. I will only refer further to accidents arising from fires and irruptions of water. Underground fires are of two classes, those due to spontaneous combustion, called gob-fires, and others caused by accidental ignitions. Fires of either class may result in loss of life either by suffocation by the products of combustion or by explosions caused by the fires. In seams free from fire-damp gob-fires do not usually have fatal results, but in fiery and dusty seams they are a source of great danger on account of the risk of explosions. The causes of spontaneous combustion in mines are not clearly understood, and the liability of any seam to its occurrence is only discovered by experience. In fiery seams liable to gob-fires it is important that the workings should be so laid out that a fire occurring in any part of them may be shut off from

access of air with as little difficulty and in as short a time as possible, and, further, the stoppings isolating the fire should be strengthened by stowing or otherwise so as to be capable of resisting the force of an internal explosion. Constant watchfulness should also be exercised in such mines in order to discover incipient gob-fires. During recent years there have been several very serious accidents from fires underground caused by accidental ignitions, most of which were due to the use of naked lights in dry workings, and especially by "torch" or "comet" lamps, a form of lamp which should not be taken underground. Special care should be taken in the use of easily inflammable mineral oils underground. The perfection of the various ventilating-fans, and the facilities now offered for the transmission of power, render it unnecessary to have ventilating-furnaces or boiler-fires underground. The possibility of fires due to friction of machinery, etc., should be borne in mind, and means for the speedy extinction of fires should be provided at places where it appears possible that a fire might occur.

Irruptions of water, causing loss of life, may occur by water bursting in from the surface when mineral is being excavated under large bodies of water, or from holing into old workings containing water. Inundations from the sudden inrush of surface-water are rare, and may be avoided by a knowledge of the nature of the strata and the exercise of a sound judgment as to the amount of cover required or of support to be left in the mine. Inundations due to holing into old workings, although not of frequent occurrence, sometimes lead to a large loss of life. They appear to be chiefly due to reliance on inaccurate or incomplete plans. Even when the plans of the drowned workings are fairly correct and the workings approaching them are bored, it is a critical operation to tap old workings under a heavy water-pressure; but when the old plans are incorrect or misleading they may be worse than useless. This leads to reflexion on the importance of keeping accurate plans of underground workings. The work of surveying mines and making the plans has in the past often been confided to incompetent hands, and even now the importance of the subject is frequently underrated. It would not appear to be unreasonable to insist that persons entrusted with the making of mining plans should be required by law to hold some recognized proof of their capability.

It is usually in connexion with fires, explosions, and inundations that the vital importance of having an easily accessible second outlet for the workmen appears. Means should be taken to make the workmen, or a considerable proportion of them, acquainted with the roads leading to the

second outlet, and in complicated routes guide-boards should be placed to direct them. In cases where the outlet-shaft is not in ordinary use every possible preparation should be made, so as to reduce the time required for getting ready and winding men.

Surface Accidents.—Accidents on the surface in connexion with mines amount to 7½ per cent. of all the accidents recorded since 1851, and for the last twenty years at least the statistics show no improvement, the average death-rate being slightly under 1 per 1,000 of the persons employed above ground. These accidents are not of a character peculiar to mining, but are similar to those occurring at other works where machinery and boilers are used, and where railway-waggons are moved in confined places. The number of accidents in connexion with the movement of waggons about the screens might be reduced by the provision of greater clearance between the waggons and the structures near which they are moved.

I have not attempted to deal exhaustively with my theme, which would require a volume, but I trust this brief review of some questions relating to safety in mines has not been without interest.

Training of Mining Engineers.—In view of the continual development of the mining industry, and the increased difficulties and dangers to be overcome as mines become deeper and more extensive, and the introduction of complicated and costly machinery of various kinds, it is more than ever requisite that, to be successful, the mining engineer should have a very efficient training in both the practical and scientific knowledge of his profession. There is no reason why a thoroughly practical knowledge of mining should not be accompanied by considerable scientific attainments, and the mining engineer will find it advantageous to have some acquaintance with nearly all the exact and physical sciences. It is true that no amount of purely scientific or theoretical knowledge would fit a man to manage even a small colliery in the absence of practical experience, but the capability of the practical man is greatly increased by scientific knowledge, and whatever developments in mining may occur in the future are likely to be in a direction where such knowledge will be more than ever necessary.

The facilities now afforded for sound scientific teaching, chiefly by the establishment of colleges of science and technical schools, are much greater than they were a few years ago, although even now the equipment of

the country in this respect is, in my opinion, far from complete. By the operation of the Technical Instruction Act, 1889, there is now in some districts better provision for the technical instruction of workmen and under officials than for the scientific training of those who, in the future, will direct the mines where the workmen are employed. To maintain our industrial position we should be able to take early advantage of such scientific discoveries and improvements as will promote the safe and economical working of mines.

The competition of foreign countries is constantly increasing, and the output of coal in the principal foreign coal-producing countries increases at a more rapid rate than our own. I have not been able to ascertain the output of coal in the United States of America for last year, and 1893 does not afford a fair comparison on account of the great wages dispute in England in that year, but the coal-production of the United States progresses so rapidly that it will probably soon be greater than that of the United Kingdom. The production of coal by our European competitors is also increasing at a more rapid rate than ours, as is shown by the following table :—

Countries.	Output of Coal in 1884.	Output of Coal in 1894.	Increase in 1894 over 1884.	Percentage of increase in 1894 over 1884.
	Tons.	Tons.	Tons.	Per Cent.
United Kingdom	160,758,000	188,277,000	27,519,000	17·1
Germany, France, and Belgium ...	93,288,000	122,297,000	29,009,000	31·1

I would therefore urge all young men studying mining to endeavour to obtain scientific as well as practical training in the days of their pupilage, because if the foundation is not then laid it is difficult to acquire it in after years. Apart from the material advantage to be derived from scientific knowledge, it trains the intellect, and often develops a taste for some special branch, the study of which, instead of being a task, becomes a source of absorbing interest and unalloyed pleasure.

Distribution of Power by Electricity.—A subject of interest to colliery owners and mining engineers which appears to be looming in the not very distant future, is the establishment of large central installations situated in coal-fields, for the distribution of power to distant places by electricity. If this be found practicable there will be an immense saving in the cost of carriage of coal to the consumer, and a miti-

tion of the smoke-plague afflicting so many manufacturing towns. In some cases it might even be to the advantage of colliery owners to initiate the system for the distribution of energy to groups of mines.

Interior Heat of the Earth.—A much more speculative question refers to the possibility of utilizing the interior heat of the earth by means of deep shafts and boreholes. A few weeks ago I read of a scheme for attempting to solve this question by a series of shafts and borings in connexion with the Paris Exhibition of 1900. If this source of heat and power ever became available on a large scale it might materially reduce the demand for both miners and mining engineers; hence there is solace in the reflexion that many things are likely to happen before we get deep enough to be able utilize the heat of the nether regions.

Mr. Emerson Bainbridge (Sheffield) said he had much pleasure in proposing that the best thanks of this Institution be given to the President for the address which they had just listened to. Mr. Atkinson, from his experience as one of H.M. Inspectors of Mines, was well qualified to speak on the subject of accidents in mines. The decrease in the death-rate from explosions per 1,000 from 0·992 for the years 1851-60 to 0·237 in the years 1891-94, proved the benefits of the combined action of legislation and of the care of colliery owners and mining engineers in reducing accidents in mines. There was a danger of legislation going too far, and the endeavour, very often on trivial points, to promote still further safety became a serious obstacle to the profitable working of coal. Up to the present time, perhaps, there had been nothing wrong in that respect, and all had done their best both to initiate and carry out legislation for the improvement of safety in mines. The President had referred in his address to the Royal Commission on Explosions from Coal-dust in Mines, of which he had the honour to be a member. There seemed to be a generally mistaken idea as to the recommendations of that Commission. It went little further than to ask the owners of coal-mines to enforce existing regulations, except in the case of those coal-mines which were emphatically dangerous. He hoped the Coal-mines Regulation Bill would be carefully considered by colliery proprietors, mining engineers, and workmen so that it might not cause serious injury to the production of coal in the future. He would like to suggest one or two points to Mr. Atkinson, upon which the latter might

perhaps give them some further information. The first was perhaps rather difficult, namely, to make a distinction between the number of accidents which occurred through want of precautions on the part of the manager as compared with the number which occurred in consequence of the neglect of the persons injured. The other suggestion was that not only the statistics as to accidents in the United Kingdom, but also those of foreign countries should be given. This comparison would be extremely interesting, especially with regard to the United States of America, Germany, and France, although in many ways, British mines were more dangerous—and the comparison, therefore, would not be in all respects fair.

Mr. JEREMIAH HEAD (London) had great pleasure in seconding the proposal for a vote of thanks to the President. The address was a most able one, dealing with a subject on which the President was thoroughly competent to speak, and he was sure that the observations would receive everywhere the attention that they deserved. Mr. Bainbridge had referred to one of the leading subjects of the address, namely, the endeavour to secure the greater safety of life among the miners. He might, perhaps, be allowed to allude to the later portion of the address, in which the President spoke of the more rapid rate at which collieries in foreign lands were competing with this country. This was a serious matter for reflexion, knowing that our industrial progress and success so largely depended on the efficiency with which coal could be got out of the earth. They all knew that at the present time the cost of getting out the coal was greater than it had been in the past, and this was a matter to which mining engineers should devote their attention, not only in the interests of the colliery proprietors, but in the interests of the workmen themselves —large numbers of whom were unemployed—and in the interests of the industries of the country, which in almost every case depended largely upon cheap coal.

The vote of thanks was carried with acclamation.

The PRESIDENT thanked the meeting for the very kind way in which they had received his presidential address. With regard to Mr. Bainbridge's remarks as to the possibility of separating accidents due to mismanagement or causes within the control of the management, from those due to the neglect of the persons killed or injured, he saw very great difficulties in arriving at any satisfactory statistics; in many cases, especially in regard to large accidents, it was often a matter of dispute as to what was actually the cause and what led up to it. With respect to smaller accidents, it was often a matter of opinion; and in other cases,

it was difficult to ascertain exactly the relative importance of the causes that led to the accident. He thought that Mr. Bainbridge's suggestion was a very good one, as to the comparison of the death-rate from accidents in mines at home, compared with foreign countries; and although he was not in a position to follow it up himself, he thought that it might be taken up by any member of the Institution having sufficient time to devote to the question.

Mr. T. CLARKSON read the following paper on "Sampling":—

SAMPLING.

By T. CLARKSON, Assoc. M. Inst. C.E.

In these days of competition, when the unchangeable law of the survival of the fittest is continually being illustrated in a forcible and serious manner, it behoves every practical engineer to look closely into minute details, which have been hitherto regarded as unimportant. The days of approximation, rule-of-thumb, and guesswork are passing rapidly away, and in many departments of engineering (using the word in its widest sense) the constant demand is for appliances of greater exactness and precision. Unfortunately, some engineers still imagine that exactness is a luxury of no commercial value, and therefore an unnecessary expense. Referring to that "luxury," Dr. John Hopkinson, at the recent meeting of the Institution of Mechanical Engineers, said that "it should be borne in mind that in engineering progress, the luxuries of to-day become the necessities of to-morrow." Instead of adding to the expense, the only way to obtain true economy is to make use of the best means and tools made available by engineering science.

One of the details which concerns mining engineers, and which has not hitherto received the attention it merits, is the important operation of sampling. This work is frequently done in a most perfunctory and slip-shod manner, although so much that is of value depends upon the result. Surely sampling is co-equal in importance with chemical analysis, being in fact, the first practical step in that analytical operation.

The practical utility and value of chemical analysis is generally appreciated, and it is therefore accorded a high and respected position; but before a chemist can make an analysis he needs a small quantity of the substance from the large bulk whose value it is the purpose and object of his analysis to determine. The work of selecting this sample is often entrusted to an unskilled labourer, or sometimes it is left in the hands of a skilled but interested party. After the sample has been taken, the careful work of the analyst, who patiently gathers up all the minute fragments and weighs them to 0·001 grain, can only be compared for stupidity to locking a stable-door after the horse is stolen. Every one who has considered the matter knows that, to select a sample, which in

truth represents and embodies all the qualities and elements in their fair and correct proportions as they exist in the bulk of material, is a matter which demands extreme care, and even then the result has not always been quite satisfactory.

If the result of the analysis is to be entitled to respect, it should be recognized that not only must the latter part of the analytical operation be carefully conducted, but the beginning, which is the very foundation, must receive equal, if not greater attention. If materials were uniform in constitution, sampling would then be one of the simplest things in the world, but we know that in a heap of ore or coal the first piece that is picked up may be practically worthless, and the next may be of extra good quality, and so on ; it also generally happens that the fine stuff, that is, the dust, is very different in composition from the lumps. For example, in many silver ores the richest parts are so brittle and friable that in the process of handling they are crushed and broken to a greater extent than the harder rocky matrix, consequently the fine dust in such cases contains the best and most valuable portions of the ore. For all minerals, therefore, it is necessary to obtain in the sample the same proportion of dust and finer material as in the whole bulk.

Probably the want of expeditious means of accurately taking samples has caused reliance to be placed upon guesswork and rough approximation. As a practical example of modern (that ought soon to be ancient) practice, a maker contracts to supply Portland cement, in which there must not be more than 10 per cent. residue over a 70 mesh screen. The cement is generally never sifted nor sampled in bulk ; and the maker does not know whether there is 1 or 20 per cent. of residue ; if the former, he is wasting money in producing too high a quality ; and if the latter, he stands to lose what is more valuable than money, his reputation. In another case, makers of furnace-coke put down a costly plant to wash the coal, and eliminate as much of the incombustible parts as possible ; but there is one part of the plant still missing—and that is an easy means of exactly sampling the whole bulk of the washed coal in order to ascertain what is the true average proportion of ash left in the coal. If a maker possesses exact knowledge of the quality which he can with safety guarantee to supply in bulk, his business position must be considerably strengthened, and he has a material advantage in making a bargain, for, after all our scientific work and technical experience, it is the man who can make the best bargain who most surely succeeds.

Another point touches upon the shady side of human nature. There are some samplers who—for their own reasons—prefer the chance of

mistakes to a reliable system which practically precludes the possibility of error. The reasons for this preference can be better imagined than described, but it has been said that "if there was a perfect system of sampling it would not afford an opportunity for the display of intelligence." A professional sampler, whose skill in sampling ores was both admirable and abominable, stated last year with much satisfaction that during the last twenty years he had earned for his employers about £5,000 or, to more truthfully paraphrase his statement, certain people had lost that sum. His superior skill may generally be used on behalf of his employers, but the position also affords facilities for practice in the opposite direction.

The writer is of opinion that generally the only way to correctly sample a large bulk of material is to deal with the whole bulk, and not as at present with only a small percentage of it. To deal with the whole bulk is not practicable by hand-labour, therefore it is necessary to employ machinery. The employment of machinery for sampling has the great advantage of the work being done impartially, the results are obtained more quickly, and, most important of all, the utmost possible accuracy is secured.

It is not the purpose of this paper to recommend the employment of any particular sampling-machine; suffice to say that efficient machines are now available. Several kinds of sampling-machines are in use in America, but the one shown in the model exhibited is, the writer believes, the only one in use in this country.

The object of this little paper is to induce the members to consider a subject of great importance, one in which any improvement is obviously to the ultimate advantage of all.

Mr. JOSEPH GARLAND (London) wrote that the subject of sampling covered more ground than the author had dealt with in his short but suggestive paper. One would like, for instance, to have had a few practical hints from him as to how to take an accurate sample from a mineral-vein underground; and how to ensure a correct sample being taken from a heap of ore when uncrushed, and in lumps. It was apparent, however, that the author was dealing with crushed or pulverized material, and every practical engineer would probably agree with his contention that sampling was a very important operation, ranking, indeed, in importance with chemical analysis itself; for it was obvious

that an analysis, no matter how scientifically exact it might be, had no value, unless the sample fairly represented the bulk. Many examples could be adduced of cases where two or more analyses of samples from the same pile of crushed ore were found to disagree, though made by chemists of the highest repute, and recognized reliability, and where the discrepancies could only be accounted for on the assumption that accurate samples had not been taken. It was, however, as the author pointed out, not an easy matter to select a true representative sample without the employment of a machine. But it must be admitted that the employment of a machine-sampler was not universally applicable, and that it often happened that hand-labour only was practicable. For example, it was difficult to see how a machine-sampler could deal with large quantities of accumulated ore, or what better method could be adopted than that employed in the Cornish mines, where piles of ore of, say, 50 or 60 tons have to be fairly sampled in the interest of the mine and of the smelter. The Cornish method of sampling a parcel of ore in the days when copper was largely produced in Cornwall was first to mix the ore thoroughly by turning the heap completely over. The parcel was next divided into six equal and distinct doles or heaps by means of handbarrows of equal capacity. On the sampling-day, which took place once a month or once in two months, the sampler (the purchasing company's agent) would come to the mine and take the sample, upon the assay of which the bid or price to be offered for the parcel was to be based. Selecting at his discretion one, or perhaps two, of the doles, if the parcel happened to be large, he would direct that a cut or pathway should be made right through the pile. He would then trim down from the sides of the cutting, say, from 1 cwt. to 2 cwts. of the ore, which would be taken to the sampling-house and there be repeatedly mixed and sub-divided on a smooth cast-iron plate till only a few pounds were left. This was then pounded down by bucking-hammers until the whole would pass through a sieve having eight holes to the linear inch. After a final mixing by hand, the sample was placed in a number of small bags and duly sealed, one or two being kept by the mine, and one being sent to each copper-smelting company, willing to bid for the parcel. The whole process was systematically and carefully carried out, the final operations being jealously guarded both by the agent of the mine and the representative of the purchasing company, and the system was found to work smoothly and satisfactorily. It was at the point where the 1 or 2 cwts. of ore was taken from the dole that the machine-sampler would be found exceedingly useful, and would ensure

greater accuracy than was possible by hand-sampling. The mechanical-sampler was undoubtedly an excellent contrivance where it could be adopted to treat pulverized material as it was produced. As for instance, where the ore from a crushing-mill was made to pass direct into it. This would ensure practically perfect sampling. The laboratory sampling-machine was also exceedingly useful to the assayer or analyst, who wanted to divide the sample submitted to him into several exact duplicates for check-assay, or for future testing.

Mr. A. G. CHARLETON (London) thought that the importance of the subject of sampling could scarcely be overestimated by anyone connected with ore-mining. As Mr. A. D. Hodges, Jun., has justly remarked, "Unless a company knows with certainty the value of its ores, and the actual results of its operations—and it cannot know these without accurate sampling and assaying—it must work blindly. The wonder is that so many companies can be found who are willing to risk their fortunes by such blind work for the sake of saving a trivial additional expense; but under such circumstances it is no wonder that we often hear of undertakings managed in such a way coming to grief." Mr. Glen has pointed out that "where conscious choice is permitted to enter into the operation, a fair sample will not result except by a miracle." Exact sampling is the indispensable first step towards learning the value of any boxfull, carload, or shipload of ore. If large dumps of several hundred tons capacity are likely to require sampling, it is good practice to form a small dump at the side, by tipping every tenth or twentieth barrow-load on to it. Dr. Raymond further observes "that when the capacity of crushing and sampling apparatus is adequate, and the arrangements for receiving and handling ore are so perfect as to avoid demurrage on railway cars and rehandling of the material, the crushing and mechanical sampling of all lots received, is not only better, but cheaper than hand-sampling. It is a relief when any one department of management can be placed in the category of comparative routine calling for fidelity rather than discretion; and where the sample has to be analysed for its earthy bases, so as to properly mix it for smelting, it is usually advantageous to crush it in bulk, as the gangue-material of an ore may break very differently according to its hardness or cleavage, and the lumps will be entirely different in composition from the fines. Sampling without crushing can only be done without risk, when the character of the ore is well known, when its grade is not too high, when the difference in richness between coarse and fine pieces is not great, and when careful trial has shown that sampling without preliminary crushing is sufficiently accurate, or when avoiding

the extra cost of crushing would be an important saving. The nature of the contract between the smelter and miner is also another element to be taken into account. Perhaps the ore is not paid for strictly according to sample assay, a thing which now and then happens, though not often, in the purchase of large dumps at a round price, or the treatment of ores on joint-account, to be settled by the net results instead of the preliminary assays, or under contracts in which special terms have been secured. Perhaps again the smelter owns the mine, and can thus afford to save current expense by sampling his own ore in a more rough-and-ready manner, and run certain risks he would not take with ore, which he had to pay for according to sample-assay." The brittle ores of silver, which Mr. Clarkson refers to, present the greatest difficulty to accurate sampling, and those which contain a large proportion of metallics. Oxidized material containing gold, which comes, perhaps, next on the list, is not so difficult to handle, as it has a greater tendency to pack, and lie where it falls—unless artificially dried and pulverized—and not to form flying rolling dust. Machine-samples may be taken in two radically different ways:—(1) Part of the stream of material is taken continuously; (2) The whole stream is taken at intervals. Personally, he (Mr. Charleton) preferred machines working on the latter principle, but it was not his intention to discuss the details or relative merits of different mechanical sampling-machines employed for this purpose. Hand-sampling gave as good results as could be obtained, he believed, by machine-sampling; but all hand-sampling was subject to possible incompetence, carelessness, or dishonesty on the part of the workmen employed, and was tedious and expensive, required large floor-space, and usually entailed considerable delay, besides involving elaborate precautions against error and possible "salting," which might be done intentionally or occur accidentally; unless a room could be partitioned off from the main crushing-floor, the dust which settled on so large an area (derived from different lots of ore) might "salt" a fresh pile when thinly spread over it. A new form of sampling-shovel had within the last year or two been introduced at the Taylor and Brunton works in Colorado. It consists of a flat-bottomed, well-balanced steel shovel, 10 inches wide, having vertical sides and two central partitions, $2\frac{1}{2}$ inches apart, dividing the shovel into three compartments, the outer ones being open at the back, whilst the middle one (which is a quarter of the width of the whole) is closed. The shovel is pushed into a heap, and as it is raised, it is drawn back with a sharp rotatory motion to the right, which throws the ore contained in the outside compartments out at the back on to the rejected pile. The motion is then reversed and the shovel brought rapidly round to the

left by which it discharges the sample on to the ore-pile. A few weeks' practice, it was said, would enable the operator to sample a pile of ore almost as quickly as it could be shovelled over. In some recent speed trials, it was found that a ton of ore could be cut down to 100 lbs. sample by one man in fifteen minutes. Tests at sampling-works and different smelters upon hundreds of lots of ore, made in duplicate and triplicate showed no difference between the results obtained by this method and Cornish quartering. In gold mills, where the results were regularly checked by sampling (a practice unfortunately not by any means common) a car-sample was sometimes taken as a check on the battery-sample, but this latter and the tailings-sample were the two that were recognized to be of most importance. If the amalgam caught in the boxes was kept separate from the rest of the mill-amalgam, the bullion yields and the return as indicated by the battery-sample showed the average value of the ore milled. There were various mechanical devices for taking accurate tailings-samples, but in this connexion he could not help thinking that a mechanical device might be advantageously employed for taking the battery-samples also, as their utility was frequently vitiated through carelessness or neglect on the part of the battery-man. It would, he (Mr. Charleton) thought, be no very difficult matter to introduce a sampler under the lip of the mortar, which would divert the whole of the stream of pulp at periodical intervals into a launder, from which it could be run into a covered pail at the side of the battery the contents of which could not be tampered with, and which could be relied upon as being an accurate sample of the day's run. The simplest way of effecting this purpose would probably be to have a narrow plate pivoted under the lip of the box, corresponding in length with the width of the copper-tables, which could be tripped backwards by an arrangement actuated by a train of geared wheels driven at slow speed by a belt running over a small pulley. The plate would be automatically reset, so as to carry the pulp on to the tables by means of a lever and adjustable counterweight.

Mr. T. E. FORSTER said they were indebted to Mr. Clarkson for bringing before the Institution a paper on a subject of considerable importance. It would be further interesting if the writer would give them some information about the different kinds of samplers in use in America. The sampling of thousands of tons of washed coal per day was a different matter from sampling small quantities of ore.

Mr. T. CLARKSON, in reply, said that the object of the paper was not to deal in a comprehensive manner with an important subject like sampling, but to bring before the notice of the members what he had found to

be the greatest difficulty in the way of the introduction of machine-sampling. First of all, it raised the question of machine *versus* manual labour. He confessed that his own creed was to never do by hand what he could do with equal efficiency by machine, and generally the machine could do its work the more efficiently of the two, although, as Mr. Garland wished him to admit, the employment of a machine-sampler was not universally admissible. There was no need to admit this fact, for such a claim was never made, and sampling like many other things would in some instances necessarily have to be done by hand. Sampling in the mine was one of the most difficult of operations; they must first of all have the vein exposed, and having got as much as possible exposed, the next step was to get by hand as large a portion as possible from the lode to be reduced and cut down by the machine. In taking a correct sample from a heap of uncrushed ore, one ought to try and get rid of the prevalent notion that sampling by machinery necessitated all the stuff being crushed. This was not so. In the first place, it must be remembered that there was a certain definite relationship between the size of the lump and the size of the bulk; that is, if they had six lumps the size of their heads they could not take one as a sample; the theory of it all was a subject for exhaustive mathematical investigation, but it may now be taken for granted that there was a certain relationship between the size of the largest piece and the size of the parcel. Mr. Garland had described the method employed in Cornwall for sampling copper ores, and proved the laborious nature of the operation. First, the whole bulk was turned over by hand, then divided into, say, six parcels, one or two of which were taken haphazard and as the rest was never examined at all, the accuracy of the sample entirely depended on the mixing. He (Mr. Clarkson) was not aware at the present time of any plant either at home or abroad for dealing with washed coal, but there was not the slightest difficulty in dealing with 400 or 500 tons per day. All that was necessary was to interpose such an arrangement as was indicated in his paper, where, as all the stuff went through, they could then guarantee that the bulk would be the same as that indicated by the sample taken for assay.

The PRESIDENT said the members were all very much obliged to Mr. Clarkson for his interesting paper, and had great pleasure in proposing a vote of thanks to him.

The vote of thanks was unanimously accorded.

Prof. VIVIAN B. LEWES read the following paper on "Mining Explosives":—

MINING EXPLOSIVES.

By Prof. VIVIAN B. LEWES, Royal Naval College, Greenwich.

Last winter the writer had the honour of delivering a course of Cantor lectures before the Society of Arts on the subject of "Explosives and their Modern Development," and in the last lecture of that course dealt with mining explosives, and showed, to his own satisfaction at any rate, that all explosives which give rise to carbon monoxide as a product of their combustion ought to be strictly tabooed for use in coal-mines, not only because of the risk of injury to health and life from the poisonous nature of the gas, but also because even small traces of carbon monoxide render mixtures of coal-dust and air highly explosive, a point which has, he thinks, been entirely overlooked in all experiments upon this most important subject.

Up to thirty years ago, gunpowder and a modified form known as blasting-powder were practically the only explosives used in mines, and the discovery of the detonation of nitro-glycerine by Mr. Alfred Nobel in 1864, and its introduction under the name of "Nobel's blasting-oil," marks perhaps the most important epoch in the history of blasting-explosives. After that date many and varied attempts were made to introduce new explosives which should combine the important properties of safety and efficiency. Blasting-oil soon showing its dangerous character, restrictions were placed upon its use and transport, and very shortly after its introduction attempts were made to tame its explosive properties without reducing the strength of the explosive itself, efforts in this direction giving rise to dynamite and other mixtures of this class, in which nitro-glycerine was taken up by some absorbent material and by practically converting it into a solid form did away with many of the dangers inseparable from the liquid state, and also enabled it to be more effectively detonated.

As time passed on it became manifest that there was still room for improvements in explosives for mining work, and Dr. Sprengel suggested in 1873 the possibility of mixing together hydrocarbons, built up from the elements carbon and hydrogen and containing them in a condition easily available for combustion, with highly oxidizing bodies—in such

proportions that complete combustion would be ensured. Many attempts were made to utilize this suggestion, and thirteen years after he had read his paper before the Chemical Society, several so-called safety Sprengel explosives were introduced into this country, and have proved themselves to be a considerable advance in safety and reliability over their predecessors.

In reviewing the properties of the various mining explosives now in use in coal-mines, it will be convenient to classify them according to the way in which they produce the gas which gives the explosive effect.

Class I.—Explosion due to simple combustion, as in the case of blasting-gunpowder.

Class II.—Explosion due to detonation of the whole of the explosive, as in nitro-glycerine, nitro-cotton, and some Sprengel explosives.

Class III.—Explosion due to detonation of part of the explosive and combustion of the remainder, as in carbonite, westfalit, etc.

This may at first sight seem to be an awkward and unreasonable method of classification, but inasmuch as the claims of any explosive for mining purposes must in the first place be based on its safety as regards the non-ignition of explosive mixtures in the workings of a coal-mine, and as this in turn largely depends upon the way in which the explosive generates its force, the writer prefers to adopt it in view of the considerations which he wishes to bring before the members.

The most characteristic types of the first-class are ordinary black gunpowder and blasting-powder, both mixtures of the combustibles carbon and sulphur with potassic nitrate as the oxidizing material, the great difference between the two being that whilst in ordinary gunpowder the proportions are so arranged as to give great heat-energy to the explosion, in blasting-powder a slight lowering of temperature is obtained by increasing the proportion of sulphur present, and reducing the oxidizing material, the result being that during explosion, the products of combustion although increased in volume, consist largely of imperfectly oxidized bodies which are themselves inflammable.

COMPOSITION OF POWDERS.

	Gunpowder.	Blasting-powder.		
		England.	France.	Italy.
Potassic nitrate	75	65	62	70
Sulphur	10	20	20	18
Charcoal	15	15	18	12
Totals	100	100	100	100

GASEOUS PRODUCTS OF COMBUSTION.

	Gunpowder. Fine Grain.	Mining Powder.
Carbon dioxide	50·62	32·15
Carbon monoxide	10·47	33·75
Nitrogen	33·20	19·03
Sulphuretted hydrogen	2·48	7·10
Marsh gas	0·19	2·75
Hydrogen	2·96	5·22
Oxygen	0·08	0·00
Totals	100·00	100·00

Gunpowder itself is practically never used, and the only word that can be said in favour of the blasting-powder is that it is cheap. It is absolutely unfitted for use in coal-mines, and its abolition would do away with more than three-quarters the number of deaths annually returned as being caused by mining-explosives. The great danger attending its use, however, consists in the combustible nature of the products evolved during decomposition, a factor in coal-mine explosions which the writer ventures to think cannot be overrated.

On firing a charge of 1¼ lbs. of blasting-powder, over 8 cubic feet of combustible gas, consisting chiefly of carbon monoxide, would be produced, and this when mixed with pure air, would give over 10 feet of an explosive, or at any rate rapidly burning mixture, and experiments which have been made upon the effects of fire-damp and dust combined in causing colliery explosions show conclusively that even when fire-damp is present in such minute quantities as to form a mixture very far removed from the point of explosion, it makes the mixture of coal-dust and air highly explosive. Traces of carbon monoxide will do exactly the same thing when the air is laden with coal-dust, whilst the temperature of ignition is lower than with methane, so that when the air of the mine is charged with coal-dust, the probabilities are that a very large volume of explosive mixture is formed by the rapid escape of the products of combustion into the dust-laden air, and this (being ignited either by the flame or by red-hot solid products driven out into it by a blown-out shot) initiates a considerable area of explosion.

As the explosion takes place, and as the carbon monoxide already produced is oxidized to carbon dioxide by the action upon it of water vapour present, and also by its direct combustion with oxygen, the hydrogen of the water vapour is set free, whilst the heated coal-dust also yields certain inflammable products of distillation to the air, and partial combustion of

the coal-dust gives a considerable proportion of carbon monoxide once more, and this, driven rapidly ahead of the explosion, forms, with more coal-dust and air, a new explosive zone, and so, by waves and throbs, the explosion is carried through the dust-laden galleries of the mine.

In this way any explosive which generates inflammable products of incomplete combustion is unsafe, and should never be used even in mines where fire-damp is unknown, as such explosives are quite capable of setting up an explosion with coal-dust alone.

A still greater danger arises if any trace of fire-damp exists in the mine, as this, together with dust, provides an already explosive atmosphere, whilst the products evolved by blasting-powder are capable of playing the same part as sulphur on a match, and causing ignition of the explosive mixture.

Fire-damp, as has been shown by the numerous experiments made since Sir Humphry Davy's memorable researches, is not easily inflamed, and explosive mixtures containing it not only require a temperature of over 1,200 degs. Fahr., but require this temperature to be applied for several seconds, sometimes as many as ten, before ignition takes place.

This phenomenon is due to the absolute ignition-point of methane being extremely high, far higher than the temperature at which it decomposes into hydrogen and acetylene, and the result is that at a temperature, such as 1,200 degs. Fahr., decomposition of the methane molecules first takes place, and the liberated hydrogen then igniting raises the mass to the true ignition-point of the methane. This dual action requires an appreciable time, and it is this alone which gives the comparative safety in mines where any trace of fire-damp exists.

If we take the temperature developed by the more prominent explosives, we find them to be far above the ignition-point of explosive mixtures of methane and air for a steadily applied heat.

	Degs. Cent.	Degs. Fahr.
Blasting-gelatine	3,220	5,828
Nitro-glycerine	3,170	5,738
Dynamite	2,940	5,284
Gun-cotton	2,650	4,802
Tonite	2,648	4,798·4
Picric acid	2,600	4,712
Roburite	2,100	3,812

Whilst the ignition-point of explosive mixtures of the various combustible gases which could be present in the working of the mine, either produced by the use of improper explosives, or liberated by the coal, are about :—

	Degs. Cent.	Degs. Fahr.
Methane	660 ...	1,220
Ordinary coal-gas	648 ...	1,198·4
Carbon monoxide	640 ...	1,184
Hydrogen	620 ...	1,148
Ethane	605 ...	1,121
Ethylene	580 ...	1,076
Butylene	540 ...	1,004

It is manifest, therefore, that if the products of explosion escaped into the mine at this temperature, any explosive mixture in the mine must be ignited. This temperature, however, only exists while the gases are under the pressure generated by the explosion; and directly they blow out into the workings, expansion instantly cools them below the temperature necessary to bring about the changes leading to the ignition of mixtures of methane and air, or methane, air, and coal-dust.

It is important that it should be fully realized that the factor of safety depends to a great extent upon the retarding influence of the chemical changes necessary before the ignition takes place, and it is the absence of this with explosive mixtures of other gases that constitutes a real source of danger.

Fortunately the inflammable constituent of pit-gas is practically only methane, and with the use of proper explosives, *i.e.*, explosives which can be completely detonated, and which give neither combustible products nor burning solids on explosion, a very fair degree of safety is attained.

Directly, however, inflammable gases, other than methane, are introduced, the margin of safety disappears, and with explosive mixtures which contain carbon monoxide, hydrogen, or ordinary illuminating coal-gas, the point of ignition being the true one, no time is given for the products of the explosion to cool themselves down below the ignition-point, and the gaseous mixture is fired.

It will always be noticed that in making trials with various explosives where pit-gas is used for the mixture in which the explosive is fired ignition is rare, whilst with mixtures of air and coal-gas, ignition is the rule rather than the exception; and surely no one can believe that this depends upon the few degrees higher point of ignition which the methane is supposed to possess, and it is this obliteration of the factor of retarded ignition which makes it imperative to discard any explosive generating combustible products of incomplete combustion for use in fiery coal-mines.

It is also evident that the more rapid the explosion the safer will it be, and no explosive should be used which relies upon simple combustion either as the primary or secondary principle in its action.

A still greater source of danger found in the explosion of most blasting-powders is the excess of sulphur which it contains, and which during explosion shows its presence by the evil odour of the escaping gases which contain over 7 per cent. of sulphuretted hydrogen, whilst under certain conditions traces of carbon bisulphide are also produced.

As has been already pointed out, the ignition-point of carbon monoxide is about the same as ordinary coal-gas, and may be taken as being 1,184 degs. Fahr., but the vapour of carbon bisulphide has an extremely low point of ignition, and the admixture of only 3 per cent. of its vapour with carbon monoxide lowers the igniting-point to below 400 degs. Fahr.

Blasting-powder and other explosives of the first class should unhesitatingly be discarded not only as being unsafe in use, but also as deleterious to health, the products of incomplete combustion all having a distinct toxic effect on the system.

Taking now explosives of the second class, we come to nitro-glycerine, nitro-cotton, and some of the Sprengel explosives, and the distinctive characteristic of this division is that all the members of it are capable of complete detonation, provided always that the right sort of detonator is employed.

Nitro-glycerine, which first inaugurated the modern era of high explosives and commenced its career as blasting-oil, stands apart from all other nitro compounds, owing to the fact that it contains more oxygen than is necessary to complete the oxidation of the carbon and hydrogen found in its molecule.

Nitro-glycerine		Carbon dioxide		Water		Nitric oxide		Nitrogen
$2[C_3H_5(N_3O)_3]$	=	$6(CO_2)$	+	$5(H_2O)$	+	NO	+	N_5.

The result being that it evolves no combustible products, whilst its rapidity of detonation would make it the safest and best of all the blasting-explosives—were it not for the danger inseparable from its physical condition and sensitiveness to shock.

When first introduced it achieved considerable success, as the fact of its being unaffected by moisture gave it a great advantage, whilst the rapidity of its explosion made it only necessary to prepare a borehole, partly fill it with nitro-glycerine, and then fill up the hole with water, the water forming just as good a tamping for the nitro-glycerine when fired by detonation as if the hole had been plugged with wood or metal. The use of nitro-glycerine for blasting purposes is however attended with several inconveniences, as being fluid it can only be used for downward boreholes, whilst its transport in the liquid form has given rise to many accidents; and, finally, the liquid state is not very suitable for detonation,

as the fluid yields to the sudden blow, and is often scattered instead of being completely exploded, so reducing its power and becoming a source of danger in subsequent operations.

When cautiously heated to 100 degs. Cent. it slowly evaporates, at 200 degs. Cent. it burns, and detonates at 257 degs. Cent. When a lighted match is applied to it, it burns quietly away, and when the light is removed the flame generally goes out; indeed a lighted match may be extinguished by plunging it into nitro-glycerine. It is, however, detonated by a sudden blow, or by heating it to 257 degs. Cent. Nitro-glycerine becomes solid at 4 degs. Cent., but so much depends upon the length of exposure to cold that this may happen at from 8 degs. to 11 degs. Cent., and in this condition it is comparatively inert; hence it is necessary to thaw it before use, an operation attended with considerable risk. It is stated that exposure to the direct rays of the sun will convert it into a very unstable and explosive substance, and also that the presence of ozone will sometimes cause its spontaneous decomposition.

The general instability of nitro-glycerine, the liability to freeze, and the danger in thawing extends to the mixtures in which it plays an important part, and exudation of nitro-glycerine from mixtures containing it is a danger so well realized that it is needless to dwell upon it.

Moreover, some of the best of the nitro-glycerine class of explosives, such as blasting-gelatine, are among the worst offenders as regards the evolution of combustible products of combustion, as the deficiency in oxygen of the nitro-cotton employed is not made up for by the excess present in the nitro-glycerine used.

Such smokeless powders as cordite and ballistite are of much the same character as blasting-gelatine, and in all of them we find the same evolution of combustible and poisonous products marring otherwise beautiful explosives.

PRODUCTS OF COMBUSTION OF MIXTURES OF NITRO-COTTON WITH NITRO-GLYCERINE.

	Blasting Gelatine.	Ballistite.	Cordite.
Non-combustible—			
Nitrogen	20 ...	22·2 ...	19·3
Carbon dioxide ...	34 ...	35·9 ...	24·9
Combustible—			
Carbon monoxide ...	36 ...	32·6 ...	40·3
Hydrogen	10 ...	9·0 ...	14·8
Methane	trace ...	0·3 ...	0·7

It is the nitro-cotton present in these explosives which is responsible for the bulk of the carbon monoxide, whilst in cordite the vaseline

present, by supplying extra carbon, gives a still larger quantity, although more highly nitrated cellulose is employed in making it than in the case of blasting-gelatine.

Nitro-cotton alone has from time to time been used for blasting work, but in this case we obtain the maximum amount of combustible products. The following analysis shows the products from a nitro-cotton containing 13·3 per cent. nitrogen on detonation :—*

Non-combustible—		
Nitrogen	16·9	
Carbon dioxide	22·3	
	——	39·2
Combustible—		
Carbon monoxide	45·4	
Hydrogen	14·9	
Methane...	0·5	
	——	60·8

The amount of carbon monoxide produced can be reduced by admixture with various oxidizing materials, and Prof. Harold B. Dixon's observations as to the oxidizing action of water vapour upon carbon monoxide are well illustrated by the fact that if wet and dry nitro-cotton be detonated there is a notable reduction in the quantity of carbon monoxide yielded by the wet sample, and an increase in the hydrogen, showing that the water present has been acting as an oxidizing agent.

When detonated the gaseous products of the composition are :—

	Wet Gun-cotton.		Dry Gun-cotton.
Carbon dioxide	32·24	...	24·24
Carbon monoxide	27·12	...	40·50
Hydrogen	26·74	...	20·20
Nitrogen	14·00	...	14·86
Methane	0·20	...	none
Totals	100·30	...	99·80

Several explosives have been made on the principle of mixing nitro-cotton with oxidizing materials, but the only one of these still in the market is tonite, in which the generation of carbon monoxide is reduced by mixing the nitro-cotton with mineral nitrates. Such mixtures, however, give rise to a residue of fused salts, which, if blown out into an explosive atmosphere, would be extremely liable to ignite it, and although the combustible gases evolved are reduced in quantity they are not done away with.

* Messrs. W. McNab and E. Ristori.

Besides nitro-glycerine and nitro-cotton, such of the Sprengel explosives as are capable of complete detonation come under this group. The Sprengel explosives have been largely used for blasting purposes, both abroad and in this country; those used here consist of mixtures of nitrated hydrocarbons and ammonium nitrate.

Roburite, introduced by Dr. Carl Roth, is a simple mixture of nitrate of ammonium with chlorinated meta-di-nitrobenzol. The ammonium nitrate is first dried and ground, then heated in a closed steam-jacketed vessel to a temperature of 80 degs. Cent., and the melted organic compound is added, and the whole stirred until an intimate mixture is obtained. On cooling, the yellow powder is ready for use, and is stored in air-tight canisters, or is made up into cartridges. Owing to the deliquescent nature of the ammonium nitrate, the finished explosive must be kept out of contact with the atmosphere, and for this reason the cartridges are waterproofed by dipping them in melted wax.

This mixture is not exploded by ordinary percussion, firing, or electric sparks. If a layer of the explosive is struck a heavy blow with a hammer, the portion directly receiving the blow is decomposed, owing to the heat developed, but no detonation whatever takes place, nor are those portions of the substance around the spot in any way affected, whilst if roburite be mixed with gunpowder, and the gunpowder be then ignited, the latter explodes and scatters the roburite without firing it.

Roburite can only be exploded by a specially powerful detonator, and on decomposition the gases evolved contain no combustible constituents, but consist only of carbon dioxide, water, and nitrogen, with a small trace of hydrochloric acid gas, which is at once condensed by the large volume of water vapour evolved, and gives rise to no inconvenience.

Ammonite is another explosive of this class, which is manufactured from ammonium nitrate and dinitro-naphthalene, these substances being blended in the proportions necessary to give as the products of combustion carbon dioxide, water vapour, and nitrogen, but during the decomposition taking place, probably some more complex action occurs, as small traces of ammonia can generally be detected.

Bellite consists of a mixture of dinitro-benzene with ammonium nitrate, the latter being kept rather in excess.

Securite consists of ammonium nitrate and dinitro-benzene, but from the proportion of nitrate used it is probable that carbon monoxide is produced. The cartridges are coated with nitrated resin, in order to protect them from the action of the atmosphere.

There is no doubt but that this group of explosives approach more nearly to real safety-explosives than any which have yet been introduced.

The low temperature of explosion secured by the use of ammonium nitrate, the absence of any combustible products of decomposition—except perhaps with securite—and the fact that both the oxidizing material and the combustible are capable of complete detonation with a sufficiently powerful fuze, give these explosives enormous advantages over any others to be obtained, whilst they are absolutely safe in handling.

The safety of the Sprengel explosives in handling and use is to a large extent dependent upon the fact that when the mixture of ammonium nitrate and the nitrated organic body is ignited by ordinary flame, the ammonium nitrate requires a large amount of heat for its decomposition, in order to render the oxygen which it contains available for the combustion of the carbon and hydrogen in the organic body, and the temperature of the burning substance is not sufficiently high to propagate this action throughout the mass, the result being that to cause continued combustion you must have a continuous supply of heat, or the flame first started simply dies out.

The effect of this is that in handling, such bodies are practically non-inflammable, and when they are made to explode by detonation, a more than usually powerful detonator has to be employed, so that although with nitro-glycerine mixtures a charge of 7 grains of mercuric fulminate is amply sufficient to produce detonation, such a body as roburite needs at least 15 grains. Moreover, when detonation has been produced, the amount of heat absorbed by the decomposition of the ammonium nitrate causes a very considerable lowering of the temperature of explosion.

To the writer's mind, it is an absolute *sine quâ non* that in an explosive mixture for mining work, all the constituents should be capable of detonation, and the reason for this is that under these conditions the shock of the detonator resolves both the oxidizing and combustible bodies into their respective molecules, and that these then recombine into the gaseous forms which give the explosive force, the whole action being practically instantaneous, and causing the projection of the hot products with such velocity as to give no time for the decomposition of the methane in the pit-gas, and the ignition of its constituents.

In order to obtain the requisite rapidity of explosion for ensuring safety as regards the ignition of gaseous mixtures in the pit, the reacting portions of the explosive must be in the condition of molecular division, and for blasting purposes this can only be obtained by complete detonation. It is impossible to obtain safety by any attempt at

mechanical division. An excellent example of this is to be seen in westfalit, which is said to be made by mixing 95 per cent. of ammonium nitrate with 5 per cent. of shellac or resin dissolved in alcohol, the alcohol is driven off by heat, and the mixture is ground and made up into cartridges. In this mixture the resin or shellac cannot be detonated, and the presence of the inert material necessitates the use of a No. 9 detonator, containing 2·5 to 3 grammes of fulminate, to explode the mixture, and when detonated the ammonium nitrate only is decomposed, and the simple combustion of the resinous matter by the products follows as a secondary reaction, with the result that the period of explosion is very sensibly increased, and the risk of the ignition of the pit gases becomes much greater. The resinous material undergoing combustion is also a grave source of danger, as, instead of being in a molecular state of division, the smallness of the particles is governed by the degree of fineness to which it is ground, and a blown-out shot would be accompanied by a shower of sparks of the burning resin. The fine condition into which it must be ground must also increase the troubles due to the hygroscopic nature of the ammonium nitrate. On these grounds one would expect westfalit to be the least efficient and least safe of the Sprengel explosives.

In deciding as to the relative claims of the other members of the Sprengel group, ammonium nitrate being common to all, the best will be the one in which the nitrated combustible is the most susceptible to detonation, as this reduces the chance of miss-fires or partial detonation as well as increases the rapidity of explosion, and the writer should expect the chloro-dinitro-benzol used in roburite to answer best to this requirement.

The third group of explosives consists of mixtures of the first and second groups, in which a body susceptible to detonation, and generally of an oxidizing character, is exploded and the products made to act upon a combustible.

Westfalit, which has just been contrasted with the rest of the Sprengel explosives, is an admirable example of this group, but the most important member is carbonite, which consists of a mixture of about 25 parts of nitro-glycerine, 30 parts of nitrate of potassium, 4 parts of nitrate of barium, 40 parts of wood-meal, and 1 part of carbonate of sodium. On detonation, the nitro-glycerine is decomposed and combustion of the wood-meal at the expense of some of the oxygen of the nitro-glycerine and the metallic nitrates takes place. There is no doubt that the admixture of so large a proportion of carbonaceous material reduces the temperature of the explosion, but it also makes it one of the worst

offenders as regards the generation of combustible products, and if carbonite be exploded in an experimental bomb, the escaping gases can be ignited and will burn with a characteristic carbon monoxide flame, over 40 per cent. of the products of its combustion consisting of this gas.

So far, carbonite has come out in trials and in practice in a very satisfactory manner, but a blown-out shot in a dusty mine would be quite likely to lead to an explosion, whilst the fumes must be very injurious to health.

For the reasons which the writer has brought before the members, he thinks that the selection of a safety-explosive should be based upon the following points :—

1. The explosion must be due to detonation and not to simple combustion.

2. If the explosive be a mixture, both the combustible and oxidizing material must be susceptible of detonation.

3. The products of explosion must be non-inflammable and non-poisonous.

4. The explosive must be safe in handling as well as in action, and compounds of an unstable character which are liable to change should be avoided.

5. The temperature of explosion should be as low as is compatible with rapidity of action.

The following table gives an idea of how far the explosives most in use comply with these requirements, and it will be seen that the Sprengel explosives occupy the foremost place :—

MINING EXPLOSIVES.

Name.	How Exploded.	Products of Explosion.	
		Combustible.	Non-combustible.
Gunpowder ...	Combustion	14	86
Blasting-powder ...	„	42	58
Nitro-glycerine ...	Detonation	nil	100
Nitro-cotton ...	„	61	39
Gelignite	Detonation and combustion	7	93
Carbonite	„ „	41	59
Roburite	Detonation	nil	100
Ammonite ...	„	nil	100
Bellite	„	nil	100
Securite	„	trace	100
Blasting-gelatine	„	46	54
Tonite	Detonation and combustion	8	92
Westfalit	„ „	trace	100

Given an explosive which answers to these requirements, and using electric firing with detonators containing sufficient fulminate to ensure complete detonation, accidents from explosives ought to be reduced to a minimum.

Mr. A. L. STEAVENSON (Durham) said that Prof. Lewes' paper was a storehouse of information, and it was one which the members would do well to learn by heart. The information was given in such a form that it could be referred to at any time; some of it, no doubt, could be obtained from other sources, but whenever the members wanted any information on this subject, they had only to refer to this paper. There was only one hint he could give to the writer, which was, that in speaking of the use of "Mining Explosives," he should have restricted his observations to coal-mines. In some of the Cleveland ironstone-mines, about ½ ton of blasting-powder was used per day, and no other explosive could do the required work so efficiently. He had given the manufacturers of every explosive the opportunity of testing it in the ironstone-mines under his charge, and they invariably failed to accomplish results equal to those obtained with powder. From the time these mines were opened in 1850 until now, there was not a single explosive that had been able to do the work so well. Prof. Lewes had referred to the fact that red-hot products were driven out of the shot-hole: that was undoubtedly the case when using coarse blasting-powder, particularly if the fuze (or whatever else was used to ignite the powder) got to the back of the hole through the pricker-hole, and the result was a blown-out shot, with red-hot particles of powder thrown out to a distance of 100 feet. An explosion occurred some thirty years ago in the Durham mines inspection district, and red-hot pellets came out of the shot-hole and struck a workman about 50 feet away, and there were also marks of the powder on the tubs. Mr. J. J. Atkinson (the President's father) was sceptical about it, but he was afterwards convinced that it was powder, and that the workman died from the effects of the explosion of the powder. Prof. Lewes stated that:—

> Fire-damp, as has been shown by the numerous experiments made since Sir Humphry Davy's memorable researches, is not easily inflamed, and explosive mixtures containing it not only require a temperature of over 1,200 degs. Fahr. but require this temperature to be applied for several seconds, sometimes as many as ten, before ignition takes place.
>
> This phenomenon is due to the absolute ignition-point of methane being extremely high, far higher than the temperature at which it decomposes into hydrogen and acetylene, and the result is that, at a temperature such as 1,200 degs. Fahr., decomposition of the methane molecules first takes place, and the liberated hydrogen then igniting raises the mass to the true ignition-point of the methane.

This was very instructive and useful information, and might perhaps explain how it was that in the old days when they used to go and look for gas with a candle and upon the candle they found what was known as a "top," they could go very near to the explosive point—it was probably attributable to the high igniting-point of methane. In those days there was hardly a pitman to be found, whose hands and face had not been burnt by trying for gas with a candle. He was one of the few who had used blasting-oil, and he remembered carrying a tin of the explosive with which, a fortnight or three weeks afterwards, the Sheriff of Newcastle-upon-Tyne and several other gentlemen were blown to pieces. He was very pleased to find that bellite, which was his own particular favourite, stood well in Prof. Lewes' opinion. He did not know that the Professor attributed to it any particular advantage over the other two or three explosives which he mentioned, but he was pleased to find that he was right in adopting bellite. In introducing bellite at the quarries under his charge, he had done so on account of the safety of handling, but neither there nor in the coal-mines had the workmen—whose safety depended upon it—ever given them any assistance. The workmen were ready to go to Parliament to pass Acts, and to blame the management for any accident, but he was sorry to find that they had not in this matter met their employers more readily. He hoped that by some means it might come to their ears that they were behaving ill and committing an injustice to themselves.

Mr. BENEKÉ (London) said that he must correct Prof. Lewes on one or two important points in regard to his statement as to the manufacture of westfalit, which was not in accordance with facts. Prof. Lewes had had no opportunity of examining genuine westfalit, and he could not have had the advantages of practical tests with this explosive. Mr. Orsman had stated some time ago at Newcastle-upon-Tyne that he had experimented upon westfalit. He (Mr. Beneké) might, however, state that all the westfalit in this country was under his own personal control, and he had used it exclusively for experimental purposes. Another parcel of this explosive was held under an injunction by order of the Court, restraining the consignee from dealing with it in any way; and no other westfalit had been imported into this country, a fact which the Home Office could substantiate. Prof. Lewes contradicted himself; he first said that a portion of the westfalit (viz., 5 per cent.) could not be detonated, and that simple combustion took place; yet he concludes by stating that the entire products of explosion were non-combustible. It might interest Prof. Lewes to know that, as regards the temperature developed by the

more prominent explosives, there was a vast difference between roburite and westfalit. Roburite contained from 85 to 87 parts of nitrate of ammonium and 13 to 15 parts of chloro-dinitro-benzol, and westfalit consisted of 95 parts of nitrate of ammonium and 5 parts of specially prepared resin. Now, nitrate of ammonium developed a temperature of 2,057 degs. Fahr. (1,125 degs. Cent.), and the presence of chloro-dinitro-benzol, in roburite, had increased that temperature, according to Prof. Lewes, to 3,812 degs. Fahr. (2,100 degs. Cent.). Westfalit, on the other hand, only developed a temperature of 2,687 degs. Fahr. (1,475 degs. Cent.), or 1,125 degs. Fahr. less than roburite. To this fact must be attributed the vastly greater safety of westfalit over roburite in gaseous mixtures and in dust-laden galleries.

The SECRETARY said he would like to deal with a small part of what Mr. Beneké had referred to. He had two samples of westfalit, which had been handed to him since he came into the room and which he had not yet opened. Perhaps Mr. Beneké would open them and say whether they were genuine or not.

Mr. BENEKÉ said that it would be impossible without proper analyses to say of what materials the samples exhibited by the secretary were composed. He, however, considered that it was impossible for any person to become possessed of genuine samples of westfalit.

Mr. ORSMAN (Wigan) said that Prof. Lewes, who had up to now given very great attention to the question of explosives, had put forward in this paper ideas which had never before been submitted to any society, one of the principal points being the explanation why coal-gas should be more easily ignited than pit-gas.

Mr. H. RICHARDSON HEWITT (H.M. Inspector of Mines, Derby) said that Prof. Lewes' paper was a very valuable one, coming as it did at a very opportune time, when mining engineers were wishing to adopt an explosive which would do its work properly, with a minimum risk of causing an explosion of small percentages of fire-damp and coal-dust. Personally, he was an advocate for the abolition of gunpowder from all coal-mines, and he was pleased to find that he was supported by so eminent an authority as Prof. Lewes. In mines where naked lights were used, gunpowder was dangerous because of the presence of those lights, and he could relate several cases where the careless handling of gunpowder and naked lights had caused serious injury to workmen. In mines where it was found advisable to use safety-lamps, gunpowder was not a proper explosive; if it was desirable to enclose the light, it must be dangerous to use an explosive which emitted a large volume of flame, and as some

mines could not possibly be carried on without the use of some explosive agent, safe or high explosives became a necessity. If gunpowder had been used which was chemically pure, there would be less complaint as to the volume of flame emitted, that is, if suitable and sufficient tamping had been placed over charges which were not too large to do the work required. It was to be hoped that the manufacturers of high explosives would not fall into the same error of adulteration. Prof. Lewes' paper gave the first unbiassed expert opinion as to the merits of various explosives, and in view of recent experiments it was refreshing to know that there was at least one explosive which was tolerably safe under the dusty conditions of a coal-mine. Carbonite, of course, must be condemned, as 40 per cent. of its products of combustion consisted of an explosive gas. This was to be regretted, as this explosive was the lowest, as regards strength, of the high explosives, and therefore would be the most suitable explosive to use in working coal if it were not for this vital objection. A great source of danger that was frequently overlooked was due to irregularities either in the manufacture, such as incomplete mixing, or in insufficient detonation. Prof. Berthelot had shown that the reactions due to these irregularities varied under different circumstances, such as the amount of fulminate used, or the temperature of the explosive before detonation. Nitrate of ammonium was the most suitable substance for reducing the temperature of detonation, and others were:—Alum, chloride of ammonium, carbonate of soda, and sulphate of soda; although these are not so certain, as they are only partially decomposed on ignition.

Mr. JAMES ASHWORTH (Derby) wrote that he did not agree with Prof. Lewes' statement as to the composition of miners blasting-powder. Some years ago, he had a sample analysed, and the result was as follows:—

Nitrate of potassium	65·14
Sulphur	14·12
Charcoal	19·69
Moisture	1·05
Total	100·00

This mixture would give an excess of carbonic oxide, while that recorded by Prof. Lewes would give an excess of sulphurretted hydrogen.

Mr. CROWE (London) said that he had had the advantage lately of being in Germany, where he witnessed some trials of explosives made by Mr. Winkhaus.* The results of the whole series of these experiments were summarized in the following table, where the weights given were the smallest which produced ignition:—

* *Trans. Fed. Inst.*, vol. ix., page 250.

Name of Explosive.	Coal-dust only.		Mixture of Coal-dust and Fire-damp.			Mixture of Coal-dust and Fire-damp.			Mixture of Coal-dust and Fire-damp.			Fire-damp only, without Coal-dust.		
			Weights.		Fire-damp.	Weights.		Fire-damp.	Weights.		Fire-damp.	Weights.		Fire-damp.
	Gr.	Oz.	Gr.	Oz.	%	Gr.	Oz.	%	Gr.	Oz.	%	Gr.	Oz.	%
Gelatine-dynamit	100	3·53	75	2·65	2·20	45	1·58	6·50	..	..	..	50	1·76	6·20
Gesteins-carbonit (stonite)	123	4·34	111	3·92	2·10	..	..	..	..	..	..	81¶	2·86	7·00
Sekurit	100	3·53	50	1·76	2·10	..	..	..	..	..	..	150	5·29	6·40
Wetter-dynamit ..	342	12·07	200	7·05	2·20	68	2·40	6·30	..	..	..	51	1·80	6·60
Roburit	300	10·58	152	5·36	2·40	130	4·59	5·80	..	..	..	154	5·43	6·10
Westfalit	174	6·14	300*	10·58	2·10	250	8·82	6·20	..	..	..	200	7·05	6·30
Dahmenit	294	10·37	354*	12·49	2·27	250	8·82	6·30	..	..	..	251	8·85	7·10
Progressit	333	11·75	302	10·65	2·25	..	..	..	381*	13·44	7·00	..	..	.
Dahmenit, A	No ignition.‡		No ignition.§		2·25	No ignition.‖		6·30	579†	20·43	7·20	562†	19·83	7·00

NOTE.—Gr. = grammes, and oz. = ounces.

* Repacked into non-paraffined cartridge-cases.

† Repacked into non-paraffined cartridge-cases of larger diameter, as the cannon used was not large enough to take more than 500-506 grammes with the original diameter of cartridge, that charge reaching to the mouth of the cannon.

‡ Largest shot used 17·71 ounces (502 grammes).

§ Largest shot used 17·85 ounces (506 grammes).

‖ Largest shot used 17·67 ounces (501 grammes).

¶ Smallest shot used.

In these experiments, the coal-dust was poured into the explosion chamber through a hole in the top, over a fan with two wooden blades, which mixed up the coal-dust with the fire-damp. He would like to point out how the most beautiful theoretical deductions might be upset by trifles. For instance, they all knew very well that with some of the safety-explosives, which were liable to become damp, the cartridge-cases were impregnated with paraffin or ceresin or similar substances. For instance, the weight of westfalit causing an explosion of coal-dust alone was 6·14 ounces; in the next column, the weight was found to be 10·58 ounces, and that increase appeared to be due to the fact that the original paraffined cartridge-case had been removed and superseded by a paper one. The same variation occurred with regard to the original dahmenit. Of the two trial series with progressit, he had only taken the latter, as two varieties were submitted, and the former had been discarded owing to the fumes which it created. In addition to the points, which Prof. Lewes had laid down as being those which should govern the manufacture of an ideal explosive, he thought from a practical standpoint might have been added one or two more. He himself was not able to follow Prof. Lewes in all his chemical details, but he thought in practice it was desirable to get first of all, a good fall of lump coal; a second point, was that the explosive should be powerful; and another

—which was perhaps strictly included in the two already named—was cost. It was no use producing an explosive if its price were excessive.

Mr. BENEKÉ said that in some cases Mr. Winkhaus had fired the explosive flush with the mouth of the mortar, so that a portion of the detonation of the explosive actually took place in the inflammable mixture of gases and coal-dust.

Mr. WAIN said Mr. Crowe had hit the right nail on the head when he said that what they wanted was an explosive which gave them the most round coal. It was of the highest consequence to mining engineers to produce round coal, and they all knew from their day-to-day experience the powerfully shattering action of all high explosives. An explosive was wanted that resembled gunpowder in its action, and when such an explosive, combined with the safety of the other explosives, was invented, it would speedily come into general use in gaseous and dusty mines.

A MEMBER remarked that Mr. Crowe's table derived from Mr. Winkhaus' experiments was not complete, and varied from similar tables contained in the original paper.

Mr. D. M. D. STUART (Bristol) wrote that the theory advanced in Prof. Lewes' paper was, that the danger in using an explosive agent in a mine was determined by the fact whether or not it yielded combustible products in explosive ignition. The order of safety was suggested to depend upon whether the explosion was due to detonation, simple combustion, or a combination of the two actions. Upon these theories, roburite was selected as approaching nearest to the conditions of safety; and mining-powder declared to be the most dangerous. Some colliery explosions have been proved to be coincident with blasting operations with mining-powder; and upon the scientific data of the researches of Sir F. Abel and Sir A. Noble, at least one disaster had been traced to this powder. In the absence of researches of a corresponding character with other explosives, it was impossible to decide whether or not any of them would have produced identical consequences if fired under the conditions that obtained in that disaster. Comparative conclusions may be drawn, but they cannot have the value or finality of inductions from a scientific basis. There was a disposition to select mining-powder from amongst the other explosives, and make it the scapegoat for mining calamities, and to conclude that, by excluding it from all mines, explosions would practically cease. The objection to this conclusion was the failure to show what was the essential element of danger in an explosive; and whether explosives other than mining-powder were devoid of that element,

and could produce similar consequences. Prof. Lewes met this objection with his theory that the danger was due to combustible products being produced in the explosive ignition. Mining-powder was not the only explosive that yielded such products, therefore it was not alone in its condemnation. Mining-powder was first considered, and it was surprising to find that sulphur was stated to form 20 per cent. of the compound. It would probably be sufficient here to give the analysis of the powder used at the Timsbury collieries, the scene of a recent explosion, which was as follows :—

	Per Cent.
Nitrate of potassium	72·5
Sulphur	11·2
Charcoal	15·3
Moisture	0·9
Total	100·0

It will be observed that this powder, which was associated with the explosion, contained about one half of the quantity of sulphur set forth in Prof. Lewes' table;* therefore the suggested danger he advanced in an excess of sulphur had no existence in the powder which caused that disaster. Prof. Lewes' second table was headed "Gaseous products of combustion," and appeared to convey the impression that the percentages it contained were relative to the total products of combustion.† Of course Prof. Lewes did not mean to give that idea; but it was necessary to remark that the figures were the percentage compositions by volume of the gaseous products, and represented about one-half of the total products. Upon this point he might quote the following figures :—

	Per Cent.
Weight of gaseous products	51·35
Weight of solid products	47·04
Water	1·61
Total	100·00

The relative quantities of the bodies that constituted the products of fired mining powder were :—

Weights of gaseous products—	Per Cent.	Per Cent.
Carbon dioxide	22·79	
Carbon monoxide	15·22	
Nitrogen	8·58	
Sulphuretted hydrogen	3·89	
Marsh gas	0·70	
Hydrogen	0·17	
		51·35

* *Trans. Fed. Inst.*, vol. ix., page 321. † *Ibid.*, page 322.

Weights of solid products—		
Potassium carbonate	19·45	
„ sulphate	0·28	
„ monosulphide	17·45	
„ sulphocyanate...	1·39	
„ nitrate	0·04	
Ammonium sesquicarbonate	0·84	
Sulphur	6·64	
Carbon	0·95	
		47 04
Weight of water		1·61
Total		100·00

Another table was given of the products of combustion of various explosives in common terms of total products, and classed combustible and non-combustible.* Mining-powder was stated to yield 42 per cent. of combustible and 58 per cent. of non-combustible products. The inflammable bodies named in the paper were carbon monoxide, marsh gas, and hydrogen, representing 41·72 per cent. of the gaseous products, which were only 51·35 per cent of the total products, and the combustible portion was 16·09 per cent. of the total weight. Gelignite was said to yield 7 per cent. of combustible products. Dynamite was omitted from the nitroglycerine class, which was unfortunate as comparisons were raised, and its products were non-combustible. Mining-powder, gelignite, and dynamite were believed to have caused explosions. Camerton colliery explosion had been traced to mining-powder;† Apedale colliery explosion to gelignite;† the Albion‡ colliery explosion and the National§ colliery explosion were both attributed to dynamite. The combustible products in these explosives were:—

	Per Cent.
Mining-powder	16·09 by weight.
Gelignite	7·00 by volume.
Dynamite	nil.

It was evident that they did not constitute the danger of these explosives on the occasions named, and it was significant that the one which contained no combustible product was credited with the greatest mining calamity during the last quarter of a century. The *Report* of the Proceedings of the Explosives Committee of the North of England Institute of Mining and Mechanical Engineers also afforded evidence on this question.

* *Trans. Fed. Inst.*, vol. ix., page 331.

† Prof. H. B. Dixon's Memorandum to the Royal Commission on Coal-dust.

‡ Prof. H. B. Dixon's Report on the Albion Explosion.

§ Prof. W. Galloway's evidence before the Royal Commission on Coal-dust.

Referring again to the products of combustion, it will be observed that carbonite yields 41 per cent. of combustible products, and that bellite, securite, ammonite, and roburite yield no combustible products. Upon Prof. Lewes' theory it would therefore be expected that carbonite would ignite gaseous mixtures, and that the other four would not. The contrary was the case. Table II. of the Committee's *Report** showed that carbonite failed to ignite mixtures of methane and air, though it was fired 45 times, and proved itself equal to bellite and roburite, but securite and ammonite both ignited similar mixtures. In mixtures of coal-gas and air, all these explosives produced ignitions in the following order:—

Roburite	...	...	75·0 per cent. of the shots ignited the gas.	
Securite	...	...	54·4	do. do.
Bellite...	...	...	50·0	do. do.
Ammonite	...	...	40·0	do. do.
Carbonite	...	...	22·2	do. do.

The Explosives Committee drew attention to the fact that carbonite gave a smaller proportion of ignitions, and a smaller evidence of flame than any of the other explosives. The evidence in disastrous explosions, and in experimental investigations, showed that inflammable bodies as they were found in the products of the explosives discussed, did not alone constitute the danger, and that the theory was untenable. The comparisons instituted between the explosives in Prof. Lewes' paper appeared to rest upon an inadequate basis. It was natural to suppose that an explosive which yielded inflammable products would be more dangerous than one that yielded non-combustible products. There could be no doubt that we must go deeper into the history of their combustion and detonation to discover the causes of the phenomena discussed and ascertain the essential element of danger. An explosive that yielded inflammable products in part, appeared to afford more favourable conditions for the ignition of gaseous mixtures than an explosive that produced only non-combustible products, provided that the temperatures and volumes of heat at disposal were identical. To effect comparisons, the potential heat must be ascertained as completely as it was done in mining-powder. Another feature in Prof. Lewes' paper which affected the theory of safety in detonation was the remark upon the dissipation of heat in the products of the explosive. The words were: "Directly they blow out into the workings, expansion instantly cools them below the temperature necessary to bring about the changes leading to the ignition of mi

* Page 19.

tures of methane and air, or methane, air, and coal-dust."* Further, "To the writer's mind, it is an absolute *sine quâ non* that in an explosive mixture for mining work, all the constituents should be capable of detonation, . . . the whole action being practically instantaneous, and causing the projection of the hot products with such velocity as to give no time for the decomposition of the methane in the pit-gas, and the ignition of its constituents."† The suggested dissipation of the heat of the products was evidently formed upon the behaviour of the products in a universal atmosphere; it did not apply in the circumscribed passages of a mine, otherwise no danger would arise in shot-firing, and many of the explosions that have occurred would be inexplicable. When a shot was fired in a mine, the products were either expelled through the shot-hole, or through the opening planes of rupture, at a velocity that preserved them in fan-like sheets, and in that condition they struck the roof, or floor, or sides, within a limited space, and expended their heat energy in igniting mixtures of methane and air, or distilling gases from the coal-dust, of which there was abundant evidence. The theory that detonation would produce such a velocity that methane and air would not be ignited, was for the same reason untenable. It had been shown that two detonating explosives would ignite such a confined mixture. The theory did not apply in a mine passage. The heat-energy in combustion, and in detonation, was expended in rupturing the strata, and the surplus energy was expended in the gaseous mixture, or coal-dust, that the products might be propelled into. The difference here between combustion and detonation was in the rapidity of the expenditure of the energy. It could not be lost. Whether combustion or detonation, the product of the potential heat was the same in both cases, provided that the sources were identical or equal. If the heat-energy be equal, and it must be, in instituting a comparison between the phenomena of combustion and detonation, the past history of either process was immaterial. The theory that detonation was an essential factor in a safety-explosive appeared also to be untenable. It was also claimed for nitrate of ammonium compounds that "the amount of heat absorbed by the decomposition of the ammonium nitrate causes a very considerable lowering of the temperature of explosion." The same advantage may be claimed for mining-powder, whose temperature, calculated upon the products observed after cooling, was 5,245 degs. Fahr. (2,896 Cent.), whereas the temperature in the final reactions was found to be from 3,272 degs. to 3,632 degs. Fahr. (1,800 to 2,000 Cent.), showing an absorption of heat representing a reduction of temperature of between

* *Trans. Fed. Inst.*, vol. ix., page 324. † *Ibid.*, page 329.

1,613 degs. and 1,973 degs. Fahr. (896 and 1,096 Cent.) If science could elucidate the conditions under which this absorption of heat was produced, and enlarge that absorption, mining-powder would not only remain unrivalled in the economical value of the coal it dislodged, and of the labour employed, besides first cost, but it would not be second in safety to any current explosive. He (Mr. Stuart) would venture to repeat the view he had already advanced, that the danger in explosives was the heat-energy at disposal after the disruptive effects were produced; and that this element, which was common to every explosive, was the factor to be investigated and considered, and constituted a danger, which was independent of the nature of the explosion, and to a very large extent of the products also. He (Mr. Stuart) would like to join in the expression of thanks to Prof. Lewes for his very interesting paper, and though unable to accept his theories, he felt that nothing but good could result from the discussion which he had initiated by communicating to the Institution his valuable thoughts and work upon the important subject of explosives.

Prof. LEWES said he had to thank Mr. Steavenson for calling attention to the fact that the title of the paper was a distinctly misleading one; he regretted that the title was not "Explosives for Use in Coal-mines" which would have been a better title for it. Of course when they were dealing with hard material, such as the ironstone in the Cleveland district, they must have something which would push and not explode with enormous violence, otherwise they would find that with a violent explosive the hard material acted very much in the same way as a cannon, and would blow out the high explosive instead of the high explosive blowing down the material. The steady pushing action of blasting-powder made it of great importance for that purpose, inasmuch as there were no explosive mixtures made which had the same efficiency for use in such mines as blasting-powder. If westfalit was such a superior explosive, why did the gentleman state that it was not possible to get an ounce of it? Mr. Beneké pointed out that 95 per cent. of westfalit was nitrate of ammonium and 5 per cent. was specially prepared resin or shellac. Now, neither resin nor shellac was a substance which could be detonated, unless under exceptional circumstances. He did not quite see where the point of contention arose; if it were resin or shellac it could not be detonated, and they must have a dual action. He did not say that when that dual action went on with a properly tamped shot it would give out a shower of sparks, but if it were not tamped or if they had a blown-out shot, it would always be accompanied by sparks. His view in writing this paper was to bring before them

two or three points of scientific interest. Those points of interest were first, the dual action which he imagined (and which he thought he could prove) took place when mixtures of pit-gas and air were being exploded. With a mixture of methane and air, the methane was first decomposed, and the ignition of the liberated hydrogen started the explosion. He thought that was an entirely new point, and he would be glad to have it thoroughly discussed. Another important point set forth in his paper was that the whole of the substances in any safety-explosive should be capable of detonation, and that there should not be any risk of after explosion by the evolution of combustible constituents.

Mr. A. L. STEAVENSON had great pleasure in proposing a cordial vote of thanks to Prof. Lewes for his most interesting paper.

Mr. SPENCER, in seconding the vote of thanks, said that the paper contained a large amount of valuable information.

The vote of thanks was carried with acclamation.

Mr. A. G. CHARLETON read the following paper on "The Gold-milling Process at Pestarena, Italy":—

THE GOLD-MILLING PROCESS AT PESTARENA, ITALY.

By A. G. CHARLETON.

The process adopted at the Pestarena works is of peculiar interest to mining engineers, because to a certain extent it is unique both as regards the general results that it has achieved and the details of the plant by which these results have been obtained.

The writer is not, as a rule, in favour of grinding a gold-ore, nor does he think that, except under special circumstances, dry-crushing with rolls will ever come largely into use when a gold-ore has to be dealt with which requires to be amalgamated, though this is a very vexed question; but the reasons that induce him to hold this opinion are very simple and clear:—

Where large quantities of ore have to be crushed to a size equivalent to the mesh to which the pulp would be reduced in a stamp-battery, it is generally, he believes, found cheaper to crush a gold-ore wet with stamps than dry with rolls. Even dismissing this objection to rolls, there remains the difficulty of amalgamating the ore after it has been dry-crushed, and this difficulty is a very real and tangible one. By employing a battery of wet-stamps, nothing is easier or more inexpensive than to pass the pulp over copper-plates, but if the ore is crushed dry, the use of copper-plates is altogether debarred, owing to the practical difficulty of mixing the ore when dry, so as to form a pulp of a proper consistency which can be fed in a regular stream over the copper-plates as rapidly as it is crushed. Every miner knows the time it takes to mix a mass of fine dry ore in a powdered condition with water, and that any attempt to do so will inevitably result in the formation of a large quantity of slime covered with a film of air, which never becomes wetted, and floats away.

It may be argued, however, that this objection can be got over by crushing the ore with rolls when wet, and to some extent it is so, but even then there is still a very vital obstacle to amalgamation by the copper-plate method. In a stamp-battery, the pulp is distributed along a long line of stamps, at any point of which, not only can the discharge

be adjusted, and either started or stopped at will without affecting the working of the rest of the crushing-machinery and amalgamating-tables, but the regulation of the relative quantities of ore and water passing over each table is held under control.

If, however, the ore is all crushed and delivered at one point from the narrow discharge of one machine, such as a pair of rolls—how can the necessary conditions to which the writer has alluded be satisfactorily fulfilled?

We have therefore to fall back on grinding in pans or mills as the only satisfactory solution of the difficulty of treating an ore by amalgamation which has been crushed with rolls, but in this connexion cost and efficiency have to be considered. As regards cost, there can be no question that amalgamation by grinding is far more expensive than amalgamation on copper-plates, but there may be circumstances which in some cases justify its employment, whilst in others they preclude its use.

The circumstances which justify its use appear to be when there are, as in Western Australia, high-grade ores, carrying very coarse gold, combined with a scarcity of water, and where the first problem is how to treat small amounts of rich ore rather than how best to mill large amounts of low-grade stuff, two entirely different propositions. The circumstances which preclude the use of the grinding-method are determined by the question of cost, as compared with other available methods, and by the relative extraction that it will secure. This latter circumstance is governed to a large extent by the mineral combinations present in the ore, and by the condition in which the gold exists in it, as it is very certain that different ores give very different results when subjected to grinding. In some cases, the gold is entirely free and amalgamates easily up to a high percentage, in others even a pyritic ore yields a fairly good return when ground raw, whilst in certain instances it requires to be weathered or roasted to prepare it for treatment, and again in others roasting, instead of improving matters, actually reduces the extraction.

The treatment of the ore at Pestarena is of peculiar interest, partly because it is one of the few instances in which a gold-ore has been crushed on a large scale with commercially successful results by means of rolls, and partly because, considering the pyritic character of the ore, the results as regards extraction are unusually good; therefore the question naturally arises whether this is due to the condition in which the gold exists in the ore, or to the nature of the treatment.

The writer is inclined to think that it is partly owing to both causes, and that machinery of the class that has been introduced at the Pestarena mines might advantageously be substituted for berdan and other forms of grinding-pans, where it is necessary to employ the grinding-method, for reasons that have been already pointed out.

The ore (an auriferous iron pyrites) is very quartzose, associated with arsenical pyrites and graphite, and is mixed with more or less of the country rock (a graphitic micaceous schist).

Preliminary Remarks.—Before the ore is brought to the surface it is broken in the mine to a convenient size, and separated as far as possible from sterile rock, this being the specified duty of each company of contractors, the non-observance of which is punished by a fine ranging from 5 to 25 lire. The ore is then hoisted, screened to separate the roughs from the smalls, and trammed direct to the picking-floors, *A*, (Fig. 1, Plate XIX.)

Reduction.—For reducing the lump-ore to the requisite size for the mills, a Marsden stone-breaker and Cornish rolls are employed, that broken in the stone-breaker going direct to the rolls without further handling.

The screenings, when not too poor, are trammed direct to the rolls, otherwise they are first concentrated by jigging, whilst the roughs, as before mentioned, are sorted, and the better quality trammed direct to the stone-breaker, the poorer quality being first concentrated by cobbing. The (medium-size) cobbing-ore is brought by trap-doors, *C*, on to tables, *B*, and thoroughly washed by water from *G*. After washing, the ore is picked as clean as possible from sterile rock, and carried or trammed to the stone-breaker by girls, two being employed at each table.

The fine pyrites that is washed away from the tables, owing to its high specific gravity is deposited at *D* and *E*; *D* being a separate box placed at the outlet of each table, and *E*, a canal running underneath the whole series of tables and terminating in a large cistern, which ensures the whole of the pyrites being deposited. The tables are about 6½ feet (2 metres) long by 3¼ feet (1 metre) wide, and have sides 4 inches (10 centimetres) high.

When about 20 tons of ore have passed through the crusher, the pile is levelled, and cut in quarters and a sample taken. At the end of each month these samples are thoroughly mixed, and a general sample of the month's milling is taken for ascertaining the duty of the mills.

Amalgamation.—The extraction of the gold from the ore is effected by amalgamation, in what are known as Francfort mills, an improvement on the Mexican *arrastra*. These mills consist of a circular bed of granite 2 feet (60 centimetres) high, with an outer diameter of 10 feet (3·05 metres). The bed is made in from four to eight segments, which after being placed in position and levelled, are cemented together with strong hydraulic cement. The mill is then surrounded on its outer and inner diameter by wood-tubbing, *B* (Figs. 2 and 3, Plate XIX.), that on the outside resting on the floor, whilst that on the inside rests on a shoulder, *K*, cut in the granite-bed for that purpose. The outer tubbing is somewhat larger than the bed, the intervening space, *F*, being packed tight with clay to within 1 or 2 inches (3 or 4 centimetres) of the surface of the bed. This space is then covered with cement, which thus connects the bed and tubbing, and prevents the ore and mercury from passing through the packing. The joint between the bed and the inner tubbing is also closed with cement for the same purpose.

The outer tubbing is 3¼ feet (1 metre) high, and the inner one about 16 inches (40 centimetres), and both are held together by strong wrought-iron hoops.

In the centre of the hollow space inside the bed a granite-block, *D*, is fixed for carrying the vertical shaft, *P*, the remaining space being filled in with clay or sand. On the vertical shaft, *P*, a little above the tubbing, a cast-iron centre piece, *V*, is keyed, for carrying the four arms, *M*, which project nearly to the outer diameter of the bed.

On the bed are placed four granite drags, *C*, 10 inches (25 centimetres) high, their length being such that they do not touch the cement connecting the tubbing and bed. The drags are fixed, by means of bolts wedged into them, to an iron ring, *Z*, passing over their centres, which can be raised or lowered by four vertical screws, *N*, that pass through the arms and which are fastened under the iron ring as shown in Fig. 4. Besides connecting the drags with the arms, as above described, they are fastened by chains (four of which are shown) provided with screw-swivels for tightening or loosening them. The connexion is an important matter when the bed is new and rough, but after it is worn smooth the back chains are usually left slack. When the runners are worn and become light, iron weights (Fig. 5) are placed on the ring, *Z*, attached to the vertical screws.

The gearing of each mill is provided with a coupling on the main shafting, so that they can be worked independently of each other. The mills are fed by hoppers, *R*, having a movable shoot, *S*, which can be

raised or lowered as occasion requires. Underneath the mill, a spout and launders are set, the former for collecting the mercury, *G*, and the latter, *L*, for carrying off the waste. The mercury spout is made of iron or earthenware and passes under the bed to a hole 2 inches (5 centimetres) in diameter bored through it. When the mill is at work, this hole is stopped with a plug of hemp, and when necessary the whole contents of the mill can be discharged through it into the mercury canal.

At the opposite side there is a line of holes bored in the outer tubbing, provided with plugs, and enclosed by an upright shoot, *J*, connected with the waste launder, *L*.

Process of Amalgamation.—Whilst the ore is being ground, from 5 to 10 lbs. of lime is added per ton to prevent sickening and consequent loss of mercury.

The quantity of ore treated in each mill varies with its character, and ranges from 10 to 20 cwts. per twenty-four hours, pyritous ores requiring a much longer time for amalgamation.

The mills are started with about 55 gallons (250 litres) of water, and when all are fairly in motion the ore is added in separate charges of about 45 lbs. (20 kilogrammes), sufficient time being allowed for the partial grinding of each lot before another is added. During the process of grinding the mills are kept running at full speed (about 13 revolutions per minute), this operation usually taking about ten hours to complete. The speed is then reduced to 5 revolutions, about 9 lbs. (4 kilogrammes) of mercury is added to each mill, and the amalgamation-process is continued uninterruptedly for about thirteen hours more.

The hole in the mill-bed is then opened, and the whole contents of the mill are washed into the wooden dish, *H*, where the mercury and amalgam are collected: the whole operation occupying about twenty-four hours.

Within the last two years, experiments have been tried, with a view to concentrate the mineral escaping in the mill-sludge after it had been ground. For this purpose, the coarser portion of the efflux from the mills, after being separated from the slimes, was worked up on 10 Borlase buddles, but as these did not give satisfaction, the 4 on the upper floors have been replaced by 2 automatic buddles, and the 6 erected on the lower floors are to be removed, and self-acting round tables are to be erected in their place.

A new mill-house, with 6 mills, was built last year to deal with the concentrates thus obtained, but the extraction has not come up to

expectations, and it is now proposed to give up the idea of treating the concentrates by this method, and to deal with them by the cyanide or some other form of treatment. It may therefore be inferred that a portion, if not the whole, of the gold associated with the pyrites in the Pestarena ore is not more susceptible to amalgamation than has frequently been found to be the case elsewhere, and that either a very considerable percentage of the gold in the crude-ore is free, or the large quantity of quartzose sand which is produced in the mills, when treating the crude ore, exercises an important effect on the extraction results.

It is stated in the company's report for 1893-4, that 854 tons of concentrates were in hand, and that the treatment of 40·729 tons of concentrates had given a yield of only 12 ounces 13 dwts. 18 grains of gold, or 6 dwts. 5 grains per ton. The assay value of the concentrates is not stated, but the writer believes that it amounted to about 1 ounce per ton, showing an extraction from the concentrates by grinding of less than 32 per cent.

Costs and General Remarks.—The cost of a mill complete is about £40 (1,000 lire). A bed lasts from six to ten months, and a set of 4 runners from three to four weeks, depending upon the character of the stone treated. The cost of a mill-bed placed in position is about £7 12s. (190 lire), and that of a set of runners about 9s. 6d. (12 lire.)

All the machinery is worked by hydraulic power, the power required per mill being estimated at 3 horse-power.

The Marsden (14 inches) stone-breaker and Cornish rolls (20 by 13 inches) which crush the ore to pass a sieve of 9 holes per square inch, put through about 1,000 tons of ore per month, working ten hours per day and six days per week. Two labourers are required to feed them.

Daily Wages.—Girls earn from 9½d. to 11½d. (1 to 1·20 lire); labourers, 1s. 7d. to 1s. 10d. (2 to 2·30 lire); mill-men, 2s. (2·50 lire); smiths, 2s. 6d. to 3s. 9d. (3·20 to 4·70 lire); carpenters, 2s. 6d. to 3s. 2d. (3·20 to 4·00 lire); and masons, 2s. 4d. to 2s. 6d. (3·00 to 3·20 lire).

The reduction of 1 ton of ore at the Pestarena works costs, including picking and passing through stone-breaker and rolls 2s. 9d. (3·50 lire). The amalgamation of 1 ton of ore costs, including mercury-loss, wear-and-tear of beds, gearing, etc., about 6s. 5d. (7 lire) per ton. The amount of mercury lost in milling is about 5 ounces (144 grammes) per ton of ore.

The author, in a previous paper, commenting on the Pestarena process, wrote:—*

* *Trans. Fed. Inst.*, vol. iv., page 399.

Looking at the results [obtained in 1888-89 as compared with 1889-90], the question arises as to how far the extraction in grinding [the crude ore in the mills] is effected by the fineness of the metal present entering into the bullion, the degree of concentration, and the grade of the ore, and it seems to lend confirmation to a point observed in other cases, that a low-grade ore, carrying fine metal (gold or silver), with a large percentage of gangue or sand in the concentrates, yielded in proportion a higher return than when these conditions are found reversed.

It may also happen that the loss of mercury is affected by the duty of the mills. The agents, Messrs. John Taylor & Sons, in their report on the Pestarena mines for the year 1888, stated that the duty or percentage extracted of the assay contents of the gold in the ore had increased, being 0·80 as against 0·78 last year; and the consumption of mercury had increased from 242 grammes to the ton during last year [1887] to 310 grammes per ton [in 1888].

If the loss of mercury were due, as might be imagined, to the extra baseness of the ore, one would expect the yield to fall off, whereas the contrary is the case [*i.e.*, the yield increased with an increased loss of mercury]. The writer is inclined, therefore, to think that it may be rather attributed to extra grinding of the ore, which would tend to increase both the yield of gold and loss of quicksilver.

Prof. Le Neve Foster has remarked that, on the whole, more gold is extracted in these works in winter, when the water, coming from the mountains, is comparatively cold and clear, than when it is turbid and warm in summer. In July, 1887, for instance, at Val Toppa, the duty of the mills was 70·6 per cent., and varied from September to December from 91·7 to 88·5 per cent. Though it does not, in this respect, appear to follow an invariable rule in different years, this may be accounted for by modifying causes in different seasons, [and changes in the] character of the ore and its general manipulation, as to amount of grinding, etc., and perhaps, as Dr. Foster thinks, the turbidity of the water, as well as the more rapid oxidation of the pyrites in summer, may partly account for these results.

Dr. Foster roughly measured the increase in temperature in course of grinding in a Francfort mill, and found that an average rise of about 3 degs. Fahr. took place, but on the mill-bed itself the heat must have been much greater. It is interesting to compare this measurement with the warming of the water in a stamp-battery due to the conversion of wasted energy into heat, which, though not very considerable, may be sufficient to add to the solubility of certain portions of the ore, such as partially oxidized pyrites.

Tests made by Mr. T. A. Rickard, on January 10th, 1894, at the Gover mine, Amador county, California, are stated by him to have shown the following results :—

Months.	No. of days worked.	Average No. of Mills worked daily.	Amount of Ore Reduced.			Amalgam obtained.	Bar Gold Extracted.				Produce of Bar Gold.			Standard of Bar Gold in Thousandths.		Produce of Fine Gold per English Ton of Ore.						Duty.	Mercury.	
			Wet Ore.	Dry Ore.							Per English Ton.					Determined by the Assay.			Extracted by the Mills.				Net Loss Incurred.	Loss per Metric Ton.
			Metric Tons.	Metric Tons.	English Tons.	Grms.	Grms.	Ozs	Dts	Grs	Ozs	Dts	Grs	Fine Gold.	Fine Silver.	Ozs	Dts	Grs	Ozs	Dts	Grs	Per Cent.	Grms.	Grms
							Pestare	na	Dis	tri	ct.*													
1893.																								
July	28	27·1	563·055	549·994	541·304	206,610	24,400½	784	9	..	1	9	..	746·6	247·9	1	3	16	1	1	15	91·3	192,400	349
August	29	26·0	552·971	540·369	531·831	197,960	24,156½	776	15	10	1	9	5	753·1	241·0	1	6	3	1	1	23	84·0	164,100	303
September	28	26·7	584·286	568·954	559·964	175,500	22,338½	718	6	..	1	5	15	758·6	237·0	1	3	1	..	19	11	84·9	136,300	239
October	25	27·4	486·568	472·770	465·300	129,850	18,514	595	5	18	1	5	14	765·5	230·0	1	3	5	..	19	14	84·0	125,200	264
November	23½	26·2	400·928	390·447	384·278	107,620	17,525½	563	14	4	1	9	8	768·8	227·0	1	5	17	1	2	13	87·6	107,300	377
December	25	24·9	391·551	381·920	375·885	81,270	13,369	429	19	6	1	2	20	766·7	229·5	1	..	10	..	17	12	85·7	91,100	238
1894.																								
January	26	24·8	394·513	381·716	375·684	92,080	13,435	432	..	..	1	3	..	754·5	242·0	1	..	14	..	17	8	84·2	104,300	273
February	24	21·2	324·285	316·017	311·024	82,450	9,618	309	7	6	..	19	20	732·0	263·5	..	17	7	..	14	13	84·1	89,300	282
March	26	31·5	500·340	486·127	478·446	146,240	17,764	571	3	18	1	3	20	739·5	248·0	1	..	4	..	17	15	87·4	128,500	264
April	25½	29·4	477·700	463·963	456·668	159,658	19,491	626	16	12	1	7	10	740·5	246·0	1	3	4	1	..	7	87·6	129,500	279
May	26½	25·7	460·693	448·049	440·969	173,950	21 527	692	1	12	1	11	9	744·8	249·0	1	6	4	1	3	9	89·3	109,500	248
June	25½	26·4	433·738	421·831	415·166	177,870	22,103½	710	9	18	1	14	5	745·0	249·0	1	8	..	1	5	11	90·9	110,700	261
Totals and Averages..	312	26·5	5,570·628	5,422·157	5,336·519	1,731,058	224,239½	7210	8	8	1	7	0	751·55	244·0	1	3	7	1	0	7	86·8	1,488,200	274
1893.																								
							Val Top	pa	Dis	tri	ct.													
July	31	9·9	144·658	141·461	139·225	10,713	3,045	97	18	..	..	14	1	828·0	169·0	..	..	..	..	11	15	..	37,800	267
August	30	9·6	139·489	136·418	134·262	10,563	2,800½	90	1	8	..	13	10	835·5	161·0	..	..	..	..	11	5	..	59,900	438
September	25½	9·7	118·847	115·757	113·928	8,589	2,354½	75	14	6	..	13	6	826·5	169·0	..	..	..	..	10	23	..	52,700	454
October	13½	9·6	67·037	65·074	64·045	4,380	1,216	40	..	20	..	12	12	825·8	173·0	..	..	..	..	10	8	..	31,000	461
November to June, 1894							Mills stopped																	
Totals and Averages..	100	9·7	470·031	458·710	451·460	34,245	9,446	303	14	10	..	13	10	829·4	167·0	..	..	..	..	11	3	..	181,400	395

* In December, 1893, 40·729 metric tons of concentrates were treated in the Pozzone Mills, and yielded 12 ozs. 13 dwts. 18 grs. of bar gold, or 6 dwts. 3 grs. per ton.

TEMPERATURE OF THE WATER BEFORE AND AFTER LEAVING THE BATTERIES

Time.	Before. Fahr. Degs.	After. Fahr. Degs.	Temp. of Air. Fahr. Degs.
8·30 a.m.	37·0	40·0	40
„	38·0	40·5	40
„	38·0	40·5	40
3·30 p.m.	43·0	44·0	50
„	43·0	45·0	50
„	43·5	44·5	50
„	43·0	45·0	50
4·30 p.m.	42·0	44·0	48
„	42·0	45·0	48
„	42·0	45·0	48

The rise in temperature in this case seems to have varied from 1 deg. to 3 degs. Fahr.

Table I., taken from the Pestarena Company's report for 1893-94, shows the amount of ore reduced, amalgam obtained, bar gold extracted, yield per ton, gold-fineness, duty of the mills, and loss of mercury.

The improved duty of the mills, shown by the Table I., without a correspondingly increased loss of mercury, may probably be ascribed to an increase in the average grade of the ore, a rich strike having been made at Peschiera during last year.

Great care is exercised by the management in checking the milling-results (as shown by the particulars furnished in Table I.) of which a careful record has been kept for many years.

The writer is indebted to the courtesy of Messrs. John Taylor & Sons for placing at his disposal various particulars from which the details given in this paper have been chiefly derived.

Mr. JOSEPH GARLAND (London) said that Mr. Charleton had referred to the treatment of the ore at Pestarena as one of the few instances in which a gold ore has been crushed on a large scale with commercially successful results by means of rolls. Did the author attribute the success of amalgamation in the Francfort mills to the preliminary dry crushing of the ore by Cornish rolls? He was familiar with the Pestarena process of milling, but he had never regarded this as an example of the dry

FIG. 4.

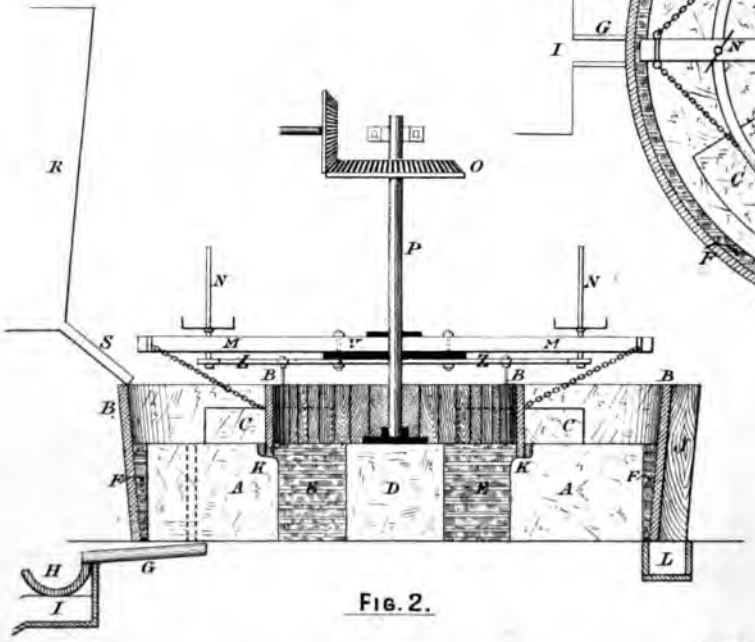


FIG. 2.

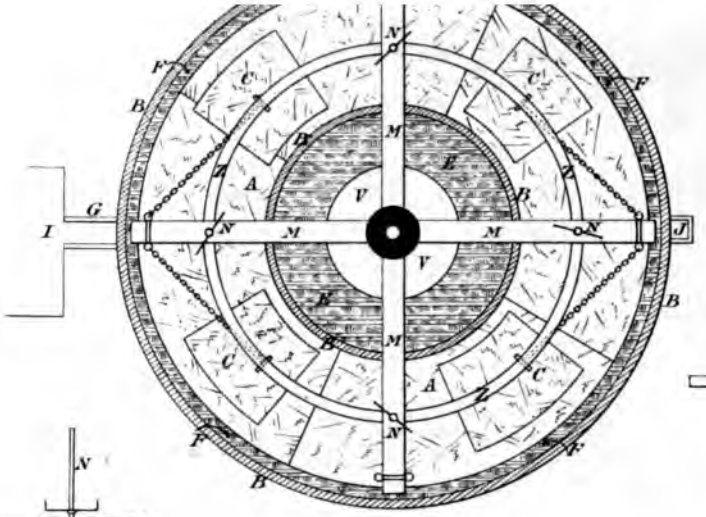


FIG. 3.

FIG. 5.

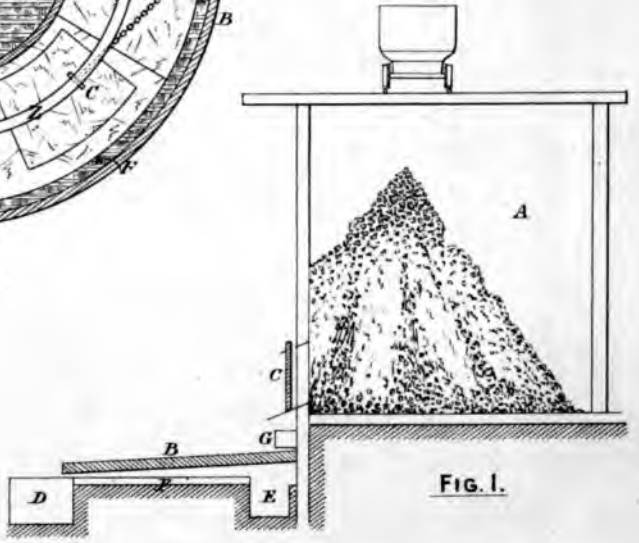


FIG. 1.

Andrew Reid & Comp.y Printers Newcastle on Tyne.

method of treating a gold ore. The fact was that Cornish rolls, like the stonebreaker, were simply an accessory, and performed the useful task of producing a partial and preliminary pulverization of the ore and nothing more. Formerly the ore was crushed by the rolls to pass through a 3 holes sieve, but for the last two or three years the rolls had been made to do a larger share of the pulverizing, and the ore was now crushed to pass through a 20 mesh sieve. From the rolls it was trammed direct to the hoppers which fed the Francfort mills, and it was in these mills that the final work of pulverizing into a pulp, liberating the particles of gold from the pyritic matrix, and amalgamation occurred. The rolls, which were a crude pulverizer, had only an accidental connexion with the success of the Francfort mills. He (Mr. Garland) did not know to what distant period the author referred when he said that "when about 20 tons of ore have passed through the crusher, the pile is levelled, and cut in quarters, and a sample taken." He thought that was ancient history. The practice now was very different. The crushed ore fell from the rolls and sieve into waggons and was trammed away direct to the mills hoppers; the sampling consisted in taking a small scoopful of the ore from each waggon. These samples were subsequently mixed, and an average sample taken as described by the author. He might also point out that the author's information was not exactly up to date in another particular. It was no longer the case that the mercury was added to the charge after the mill had been running for ten hours, and that another thirteen hours' running of the mills was necessary to complete the amalgamation. This was a fairly accurate summary of the old method, but for the last two years it had been the practice to add the mercury at the commencement of the running, so that the reduction of the ore to pulp and the amalgamation of the liberated gold went on simultaneously, the whole process requiring twelve hours for its completion. The clean-up now, therefore, took place every twelve hours. As to the theoretical consideration of the state in which the gold was held in the ore, he agreed with the author that the gold in this pyritous ore was in a free state; the fact that about 87 per cent. of the assay contents of the ore was extracted by direct amalgamation seemed to place this question beyond doubt. He complimented the author on bringing forward a paper of peculiar interest to mining engineers who were in any way connected with gold-mining.

Dr. Le Neve Foster (Llandudno) suggested that the author should record the percentage of metallic sulphides in the ores worked by the Pestarena company, as many miners will ask the question. The Francfort mill (so named after the late Dr. Francfort) is said by Mr.

Charleton to be an improvement on the Mexican *arrastra;* in reality it was a development of the old Italian *molinello* or small gold-mill. The bed and runners were not always made of granite, gneiss being largely used. He (Dr. Foster) found that gneiss beds with granite runners gave very good results. Mr. Charleton's paper lacked a section of a worn-out bed. It was not every reader who would gather from his (Mr. Charleton's) description that the runners wore a channel in the bed, and that the channel became deeper and deeper, until at last the bottom was worn through. The charge did not always require twenty-four hours to work it off; at the Val Toppa mines of the Pestarena company the charge used to be worked through in twelve hours, and required only 3 instead of 4 kilogrammes of mercury. He agreed with Mr. Charleton that the heating caused by the friction of the runner upon the bed was considerable, in fact he (Dr. Foster) stated in a paper which he had read before the British Association upon the Pestarena process more than twenty years ago, that he had seen the pyrites take fire while a mill was being cleaned out. As regarded the loss of mercury, he found in an experiment that some clean jigged ore, rich in metallic sulphides, caused a smaller loss than the original raw ore. He regretted that Mr. Charleton had omitted to give to the officials of the Pestarena company the credit which was due to them for their amalgamation-tables. For a period of more than a quarter of a century the company had regularly published tables, showing, not only the quantity of ore milled and the yield, but also the loss of gold. There were few mines where such a record existed, and a word of acknowledgment was due to the memory of the late Mr. Arthur Dean, who originated the determinations of the duty, and to Messrs. John Taylor & Sons who have continued them to the present day.

Mr. H. F. Collins (London) remarked that the percentage of gold extracted from the ore was not, in his opinion, a good criterion of the degree of perfection attained by a milling process, and that it was practically useless as a means of comparing the merits of different processes or of different appliances, unless the ore treated happened to be alike in the cases compared. A far better criterion was afforded by the absolute loss of gold per ton of ore treated. It appeared from Table I., that during the twelve months from July, 1893, to June, 1894, 86·8 per cent. of the gold contained in the ore was extracted, but that the absolute loss per ton was no less than 3 dwts. Supposing the ore had contained only 10 dwts. instead of 23 dwts., it was evident that this same loss of 3 dwts. would have corresponded with an extraction of only 70 per cent., and conversely if the ore had contained 2 ozs. no less

than 5½ dwts. might have been going away in the tailings, and yet the extraction would still appear as 86·8 per cent. of the total contents. It was evident that the possibility of effecting further economies by re-treating the tailings did not depend upon the percentage of gold which had been extracted in the original milling, but on the actual amount remaining per ton of tailings. Judging from the samples exhibited by Mr. Charleton, and on the supposition that they fairly represented the bulk of the ore treated, the Pestarena ore, though it certainly contained much pyrites and other sulphides, differed in no material respect from the ores of Gilpin County, Colorado; and these were worked in the peculiar variety of the American stamp-mill which had been developed in that district, up to an extraction of 70 per cent. on stuff which originally contained only 8 dwts., *i.e.*, on ore of this grade no less than 5·6 dwts. were extracted in the original amalgamation leaving only 2·4 dwts. in the tailings (out of which again a considerable part was recovered by concentration) as against 3 dwts. left in the Pestarena tailings. The work done therefore was fully as good from the point of view of technical perfection, while, if the cost of the two processes were compared, the comparison was not at all in favour of Pestarena. The cost was stated to be 2s. 9d. for crushing, and 6s. 5d. for amalgamation, or a total of 9s. 2d. per ton; while at the Hidden Treasure mill at Black Hawk, Gilpin County, Colorado, a large establishment under the management of his brother (Mr. A. L. Collins) running 75 stamps on ore of this character, generally called refractory, the cost had been reduced to only 3s per ton (though the cost of labour was from 10s. to 17s. per day), and even including the subsequent concentration the total cost of treating the ore did not exceed 4s. per ton. Mr. Collins would go even beyond the author of the paper in saying that there was nothing that could be usefully copied in the Pestarena process except possibly in localities like some parts of Mexico, South America, and Africa, where the following conditions obtained:—(1) High-grade ores; (2) great scarcity of water; (3) excessive cost of fuel, and (4) comparative cheapness of animal power; in which cases a modification of the process might be adopted with advantage.

Mr. CHARLETON said, with reference to the point raised by Mr. Garland as to the part which the rolls played in the Pestarena process, he did not mean it to be inferred that these were a necessary factor in its success, but what he wanted to point out was, that this was one of the few cases where rolls had been used successfully for the amalgamation treatment of gold ore. They were tried at the Haile mine, but so far they had not met there or elsewhere with the success that had resulted from the use of

stamps, largely owing to the fact that attempts had mostly been made to treat the ore after it had passed through the rolls by ordinary copper-plate methods or some modification of that system; whereas at Pestarena they were using Francfort mills; the point of peculiar interest described in his paper. Of course, indirectly, the use of rolls might be said to affect the success of the process if, as was sometimes claimed, it was cheaper to crush with rolls than with stamps: he doubted, however, that such was the case, and Mr. Collins gave some interesting figures in his presidential address to the Institute of Mining and Metallurgy which confirmed his (Mr. Charleton's) views on this point. If they were using rolls they need not reduce the ore to the same degree of fineness (as its subsequent reduction was effected in the mills), whilst with stamps it was passed at once through comparatively fine sieves, and in that way the product of the rolls being coarser, the cost of crushing was apparently reduced, and larger quantities were put through in a shorter time; but when it came to crushing to an absolute equal degree of fineness, the comparative cost seemed to be in favour of stamps. With regard to the time taken in amalgamating in the mills, he was indebted to Messrs. John Taylor & Sons for information regarding these and other details and put forward such statements as were put before him, believing them to be substantially correct, as they no doubt were at the time they were given him. The changes in the process of amalgamation to which Dr. Foster and Mr. Garland referred, were, he believed, of quite recent date, as he understood that the alteration in the time occupied in treatment had only been introduced some twelve months ago; the fact mentioned by Mr. Garland, that of recent years the rolls had been made to do a larger share of the pulverizing, no doubt admitted of running the Francfort mills a shorter time, as they had less work thrown upon them in crushing the ore; he understood, however, that since this change had been made the loss in mercury had become somewhat greater. He (Mr. Charleton) regretted that he had not got the necessary data to answer Dr. Foster's question as to the exact percentage of metallic sulphides in the ores worked by the Pestarena company. Whilst no doubt the Francfort mill was, as Dr. Foster said, a development of the old Italian *molinello*, he thought that in its general construction it presented many points of resemblance to the Mexican *arrastra*, of which it might be considered an improvement. He (Mr. Charleton) did not give a section of a worn-out bed as it would be impossible to represent in a drawing the various changes which would be constantly taking place, and he thought it was sufficiently self-evident that a channel would gradually be worn deeper and deeper in the bed by

the action of the runners. His (Mr. Charlteton's) experience, the result of numerous working trials, did not coincide with that of Dr. Foster in regard to the loss of mercury caused in grinding sulphide ores; he found that in most cases, the heavier the ore, the more chance there was of loss of mercury in grinding. Full credit was given to the officials of the Pestarena company at the conclusion of his paper for the statistics and other particulars furnished, for which he was indebted to them: this had evidently escaped Dr. Foster's notice. He was not aware that the late Mr. Arthur Dean had originated the determinations of the duty, and he had to thank Dr. Foster for bringing the matter to his attention. He (Mr. Charleton) agreed with Mr. H. F. Collins that "the possibility of effecting further economics by re-treating the tailings did not depend upon the percentage of gold which had been extracted in the original milling, but on the actual amount remaining per ton of tailings." In this case, however, the assay of the ore being given, as well as the percentage of extraction, it was easy to ascertain what the loss in question amounted to. Without submitting a sample of Colorado stone to similar treatment to that practised at Pestarena and *vice versa*, it was impossible to say with certainty how the relative extractions by the two methods would compare, and it seemed to him (Mr. Charleton) there was no evidence to show that because there was a loss at Pestarena of 3 dwts. on 23 dwts. stuff, an extraction of 70 per cent. or over could not be secured by grinding Colorado 8 dwts. stone in Francfort mills. In this case, however, the question really was whether a Colorado mill would extract 86·8 per cent. of the gold if it had to treat Pestarena stone; if it could not (whilst no one would dream of advocating the Italian process as being one that could be advantageously introduced into Colorado), in view of the local conditions at Pestarena, so long as it yielded a higher nett extraction in free gold, which would cover the extra working charges, the present practice could hold its own, despite its greater cost (about which there could be no question), as in Italy, they could not at present command the facilities which existed in Colorado for dealing with the concentrates. As he (Mr. Charleton) had himself suggested, high grade ores and great scarcity of water were two of the conditions which might justify the employment of the Pestarena process under special circumstances in countries like South America, where, as Mr. Collins said, fuel was expensive, but animal power could be substituted for it.

The PRESIDENT said he had great pleasure in moving a vote of thanks to Mr. Charleton for his paper.

Mr. A. L. STEAVENSON, in seconding the vote of thanks, said there was very little gold in this country, but at the same time the Institution had members in all parts of the world, and in their absence he was only doing what they would do if they were present in thanking Mr. Charleton for his paper.

The meeting then adjourned.

THE FEDERATED INSTITUTION OF MINING ENGINEERS.

GENERAL MEETING,

HELD IN THE ROOMS OF THE SOCIETY OF ARTS, 20, JOHN STREET, ADELPHI, LONDON, MAY 31ST, 1895.

DISCUSSION UPON MR. EDMUND SPARGO'S PAPER ON "LITHOGRAPHIC STONE AND ITS USES."*

Mr. F. W. RUDLER (London) wrote that it might be useful to point out that several complete analyses of the Solenhofen stone had been published by Dr. C. W. Von Gümbel in his *Geognostische Beschreibung des Königreichs Bayern.*† These analyses were made by Dr. A. Schwager in the laboratory of the Geological Survey of Bavaria, of which Dr. von Gümbel is the director. A blue lithographic stone, of the first quality, from Solenhofen was found to contain 98·32 per cent. of carbonate of lime and 1·68 of carbonate of magnesia. A complete analysis gave:—Lime, 53·00; magnesia, 0·77; silica, 1·17; alumina, 0·95; ferric oxide, 0·22; potash, 0·27; soda, 0·59; carbonic acid, 42·45; phosphoric acid, 0·01; water, organic matter, etc., 0·59; manganous oxide and titanic acid, traces. A yellow lithographic stone, of first quality, from Solenhofen yielded 99·34 per cent. of carbonate of lime and 0·66 of carbonate of magnesia; whilst a dark grey stone, of the best quality, gave 97·12 of carbonate of lime and 2·88 of carbonate of magnesia. The latter, therefore, approached in composition to the Croatian stone referred to in Mr. Spargo's paper.

The following "Remarks on the Banket Formation at Johannesburg, Transvaal," by Mr. A. R. Sawyer, were taken as read:—

* *Trans. Fed. Inst.*, vol. viii., page 568.

† 4th Abtheilung, "Frankenjura," 1891, page 126.

REMARKS ON THE BANKET FORMATION AT JOHANNESBURG, TRANSVAAL.

By A. R. SAWYER.

A study of the banket formation with its auriferous conglomerate-beds has a chemical as well as a geological aspect. The study of the subject in all its bearings might be considered to possess a purely academic interest, but there is no doubt that all who have the true interests of the mining industry at heart will acknowledge that it has a very high economic importance. A correct knowledge of the manner in which gold has been deposited as we now find it, and of the source whence it originated, might save much expenditure in useless prospecting and direct the miner towards payable portions of a reef. A knowledge of the geological features which obtain on the Rand, and of the nature and extent of the dykes and faults, would prevent the erroneous selection of sites for shafts, and would enable underground works to be laid out in the most advantageous manner. Disturbances which might otherwise be looked upon as injurious would be welcomed, when occurring in such a form as to enable the work to be carried out at less expense.

Reversed, or overthrust faults, which are here called overlaps—a term conveying a different meaning in a strictly geological sense—would be welcomed were it realized that they are beneficial in two ways: (1) In doubling the extent of the reef over a certain area, and (2) in reducing the depth to which deep-level shafts would otherwise have to be sunk. Nor would the presence of diorite or diabase-dykes be so entirely unwelcome, if it were found, by a widely extended examination, that gold predominated in their vicinity. A careful study of the nature of any fault passed through a shaft might, in the case of a reversed fault, prevent the error being committed of supposing that two reefs existed—from the two dislocated portions having been passed through in the shaft.

In dealing with a matter of such economic importance, a careful collection of all facts bearing on the question should precede any theories on the subject, and it should be the aim of all those who are in daily and intimate contact with the industry to note all facts bearing on the

question carefully with a view to their eventual classification. Even facts with regard to the economic extraction of gold might bear on the question as to the manner in which it has been deposited in the banket-beds.

Thus the facts lately brought out by Mr. Von Gernet in a paper which he read before the Chemical and Metallurgical Society of Johannesburg, as to the deposition of gold by electrolysis, must bear on the deposition of the precious metal in the banket beds.

In this connexion, a most interesting and apt experiment made by Mr. Andrew Crosse, sen., might be mentioned, as described in the memoir of him written by his wife :—A porous pot filled with kneaded clay was placed in a vessel of water, and funnels with various metallic solutions were placed in the clay. The clay was connected with the negative pole of a small voltaic battery, the circuit being completed by connexion with the water of the outside vessel. In the course of two years cracks were formed in the clay, and various minerals were artificially produced within the fissures. Water was also caused to circulate from the outside vessel through the fissures, and to rise to the top of the clay by the voltaic action.

It is unnecessary to repeat all that has been already written on the banket formation by the author in a paper read before the North Staffordshire Institute of Mining and Mechanical Engineers in 1889 and by numerous other writers. There are, however, some further points which require elucidation, and a few observations may be made on some more recently ascertained facts.

The banket formation proper is restricted to the Transvaal, the Orange Free State, and Zululand. It is identical with the Table Mountain Sandstone which occurs in various parts of South Africa. It covers a very large area, especially in the southern part of the Transvaal, and it is not by any means restricted to the Witwatersrand gold-field. This formation, though approaching to a basin-shape in some localities, has not so far been conclusively proved to form a real basin on the Witwatersrand. The curvature of the beds in the Witwatersrand gold-field, where these are steep at the out-crops turns to a flatter angle in depth. This curvature, which is frequently extraordinarily regular, resembling a parabolic curve, is not sufficient evidence that the strata would assume a horizontal position further south. Where the beds dip at an angle of 30 degs. and less, they do not, so far as known, assume a flatter dip in depth. Nor is the fact that these beds dip to the north in some parts of the Heidelberg district proof positive of an uninterrupted basin theory.

In connexion with a report which the author made for the Cape Government last year on the mineral resources of the Prince Albert district, a geological section was prepared from the Indian Ocean near the Knysna gold-fields to Fraserburg, covering a distance of 150 miles. That section shows in what manner the beds of that region have been tilted, crumpled, and faulted by a tangential thrust or pressure proceeding from the Indian Ocean. Numerous thrust-planes and reversed faults occur which throw the strata on to themselves, and also many anticlinals which bring up the lower strata to the surface.

It is possible that were a similar section made from Johannesburg through Heidelberg, Vryheid, and Zululand, phenomena of the same nature would be disclosed, and that it would be found that, owing to reverse faulting, the banket-beds occur nearer the surface between Johannesburg and Heidelberg than an uninterrupted basin theory would suggest.

Fig. 1 (Plate XX.) is a rough ideal section of the banket formation a few miles east of Johannesburg. It is apparent from the section that, if this view be correct, the main reef series of rocks may lie within a workable depth between their outcrop and that of the large amygdaloid quartz diabase sheet which runs east-and-west about 6 miles south, close to Mr. Meyer's house on Klipriversberg; whereas if the beds dip uninterruptedly to the south, as they would do in a basin, they would vanish below a workable depth, two miles from their outcrops.

In support of this view the author may give two instances, among a large number, which have come under his notice in the various mines on the Rand. The one occurs 10 miles east of Johannesburg, where the reef (banket bed) is thrown up 600 feet by an igneous dyke about 90 feet wide. The other occurs 12 miles west of Johannesburg, where a series of numerous small reversed faults throw the reef up over 200 feet within a very short distance.

The view illustrated in the section (Fig. 1) also admits the possibility of the same series of beds being found at one or more localities to the south-east, sufficiently near the surface to enable the payable reefs which it might contain to be worked. Such a possibility exists in the Heidelberg district over a very large area, which is in part covered by a not particularly thick coal formation.

The geological aspect of the question resolves itself into three parts: (1) The manner in which the group of rocks known as the banket formation was formed; (2) the manner in which it has become broken up, tilted, and re-adjusted; and (3) its relation to other rock-formations in South Africa, and its relative age.

The chemical aspect details the manner in which the gold was deposited in the matrix of some of the banket-beds, either at the time of their formation, or, as the author believes, subsequently.

With regard to the manner in which the banket was formed, there is no reason for supposing that the sandstones, quartzites, shales, and slates of which it is composed, are formed differently from rocks of a similar nature elsewhere, and that they are sea-borne detritus of rocks of former ages. Various theories have been advanced to explain the presence of such continuous and persistent conglomerate-beds as occur in the formation.

The writer has stated elsewhere that the "thin beds of conglomerate were probably shingle-banks or gravel-beaches as we now see them. The manner in which thick beds of conglomerate, like those occurring in the Old Red Sandstone in Scotland, were formed is not so easy of explanation. Conglomerate-beds are, as a rule, very variable in thickness; they also run in and out pretty freely. The constancy, therefore, of the banket-beds over such a great distance is remarkable. The length of their extension justly suggests a continuation downwards for a considerable distance." In thus speaking of the constancy of these banket-beds, the author had the south reef and its leaders more especially in view, which may be said to be continuous over a considerable area.

The other reefs are more lenticular, and not very different from ordinary conglomerate-beds. It is, in the writer's opinion, premature to speak of any of the theories which have so far been advanced with any degree of certainty. No satisfactory theory has yet been promulgated to account for the occurrence of the well-known Dwyka Conglomerate in Cape Colony, Natal, and Zululand; and some time must yet elapse, and we must be in possession of many more systematically collected facts before any really tangible theory can be advanced which will be generally acceptable to account for their formation.

The manner in which the banket formation has been broken up, tilted, and re-adjusted has, to some extent, already been explained. Assuming the Table Mountain and the Witwatersrand series to be identical, this formation spread at one time over the whole of South Africa; it lay horizontal and attained a thickness of about 10,000 feet. Remnants of this series are still horizontal, notably near Cape Town. The mountains from which the detritus was derived to enable such an immense mass of rocks to be formed probably occurred in the Indian Ocean. These subsided, and in subsiding produced an enormous lateral thrust from the south or south-east which affected these horizontal beds, as shown in the

writer's section of the Prince Albert goldfield. This thrust produced anticlinals and synclinals, which are very numerous in the primary series. Denudation set in and exposed the basement rocks down to the granite, which, being in a plastic condition, also felt the influence of this great thrust, being thereby forced up and increasing the general disturbance, and forcing some of the beds out of line. So great was the pressure that the beds broke and became reversed, causing the reefs to be forced nearer the surface (than they otherwise would have been) in successive steps, and this will enable them to be worked over a considerably larger area than otherwise would have been the case. In proof of this view, the author may state that 75 per cent. of the faults occuring on the Rand are reversed faults.

Eruptions of basic igneous rocks occurred previous to, concurrently with, and subsequently to the disturbance produced by a lateral thrust. The rocks forming the banket formation have not always been in the state of consistency in which they are now found. Heat and moisture combined, before they were raised to their present horizon, to form them into a pasty mass. The intrusions of igneous rocks subjected them to high temperature, and the tilting, collapse, and re-adjustment also developed heat within them.

A good example of one dyke intruded when the formation was still in a horizontal position and thrown out by one which had been intruded subsequently, occurs at the Ferreira mine.

The dykes and bosses occurring in the undisturbed coal-formation, which unconformably overlies the banket formation in places, were intruded after the banket had assumed its present position.

Diabase, an altered dolerite, predominates as the substance of these dykes. In the Heidelberg district more especially, the well-known amygdaloidal trap, similar to the rock which occurs at Klipriversberg an ancient lava in which the amygdules contain quartz, also occurs. The quartz which fills these amygdules bears some resemblance to the pebbles in some of the banket-beds. Hornblende- and chlorite-schists formed by alterations of these igneous rocks also occur, apparently interstratified with the sedimentary beds.

Although dykes on the Witwatersrand crop up in nearly all directions, the more prominent ones appear, as far as they have been observed, to run either nearly north-and-south or east-and-west. The writer does not know to what extent the dykes have been assayed for gold, but many of them contain pyrites. The so-called spar-veins, which are quartz veins occurring in the formation, also contain pyrites. Such veins sometimes occur between the reef and its footwall, as at Knights.

As regards the age of the banket formation and its chronological order, it has been demonstrated that it is younger than the well-known crystalline schists of the country, and older than the so-called "Elephant Rock," a dolomitic limestone widely distributed in the Transvaal, and the coal-formation overlying it. The writer pointed out in his report to the Cape Government that the sandstones, quartzites, and talcose slates of the Zwarteberg mountains resemble those of the Witwatersrand, and that this similarity is increased by the presence in the Zwarteberg sandstones of white, pale-red, and black quartz-pebbles, some of which are very brittle. These pebbles lie mostly scattered in indefinite lines, but the writer has found them more concentrated in the shape of thin banket-beds. The Zwarteberg rocks are, however, younger than the dolomitic limestone formation.

No fossils have, to the writer's knowledge, been found in the Witwatersrand formation. The dendritic impressions so often thought to be of organic origin are merely the result of a re-arrangement of particles, possibly assisted by a weak electric current. In corroboration of this view, the author may refer to a similar dendritic growth which was accidentally produced on H.M.S. "Rodney" last November.* Along the after-bridge, two electric light leads were fixed; the insulation of these conductors had given way, and thus an earth was established. Sea-water spray had been thrown over the leads repeatedly for a couple of days. The current managed to find its way through the white-lead paint which covered the boards of the bridge, and this paint, when so decomposed, produced a curious arborescent growth of lead 21 inches long. Apparently the carbonate of lead of the paint had been reduced to metallic lead, which assumed a dendritic shape by the electrolytic action.

The writer was fortunate enough to come across the Brachiopods *Spirifer* and *Terebratula* whilst examining the rocks in the Zwarteberg mountains. This is evidence of their being of Carboniferous age, and judging by analogy, the Witwatersrand formation will probably have been formed during the Devonian age. The writer has found what has so far been considered as true Carboniferous fossils *Lepidodendron* and *Favularia* in the more recently formed coal-formation at Vereeniging; but as many fossils of Triassic age are found in these measures, it is a question whether these "Carboniferous" fossils did not live into the Triassic age. These coal deposits lie, like all those in the Transvaal, unconformably on the banket formation, and are there-

**Engineering*, 1894.

fore much younger. The writer may remark that the beds in the Zwarteberg mountains are plicated and faulted in a remarkable manner, but are evidently very rarely traversed by igneous dykes.

With regard to the chemical aspect of the question, the author does not regard the term "pseudomorphic" (a term used by a recent writer on the subject), as applied to the matrix of the conglomerate, a happy one. A "pseudomorph" is a mineral which has replaced another, and has assumed the external form of the mineral so replaced. The writer is not aware that any mineral now found in the matrix of the banket assumes the external form of one which it has replaced. That there has been a replacement of minerals by others, in the matrix of the banket, has never been doubted. In the author's paper of 1889 he stated that:—"Rocks are porous, conglomerates are more porous than the encasing rocks, hence that would be the path chosen by the ascending hot waters or vapours containing gold, and it would be deposited in the matrix of the conglomerates, and to some extent in the pores of the encasing rocks as well."

That the conglomerate-beds are more porous, and would afford a freer road to ascending solutions, will be evident from the fact that even in cyanide-vats solutions pass more freely through the coarse material. The fact that frequently the larger the pebbles the richer the reef will also be thus explained.

The following passage is taken from the author's recent book on *The Goldfields of Mashonaland*:—"Intrusions of basic igneous rocks occur as dykes and sheets. At their junction with the schistose rocks minerals have been developed in the latter by contact-metamorphism. These being the deep-seated rocks, they frequently contain the heavier metals in greater abundance, and it is in rocks in contact with them or derived from them that mineralized veins most frequently occur. The ore particles and veinstone of which these veins are composed are derived from the neighbouring rocks, and have been deposited from solutions, not necessarily in fissures, but more often along certain cleavage or foliation-planes, by interchange or replacement of the more soluble portions of the rock."

These remarks, so far as the facts collected up to the present enable one to form an opinion, apply partly to the introduction of the gold in the banket formation. Just as in an ordinary quartz-vein there has been an interchange and replacement of the more soluble portion of the rock in which the vein was formed, so in the banket-beds, the matrix has become altered, and gold, silver, and pyrites have been carried there in

solution. The matrix has also become very much more siliceous, so that, if it were not for the pebbles, a piece of banket in depth would resemble a piece of ordinary vein-quartz.

The writer is informed by Mr. Crosse (assayer to the Standard Bank at Johannesburg) that the proportion between gold and silver, commonly called the fineness, varies greatly at different parts of the field, but that it has not yet been proved that it varies in the same locality in depth. That statement, taken in conjunction with the fact that many of the large and important dykes run north-and-south, points to the probability that the solutions carrying gold and silver rose from below upwards, so that the rocks from which the solutions derived their minerals may not yet have been disclosed in depth. This statement confirms the writer's view, expressed in 1889, that the solutions ascended from which the gold was deposited. The extreme relative proportions of gold and silver were respectively 93 per cent. of gold to 7 per cent. of silver, and 83 per cent. of gold to 17 per cent. of silver.

It is stated as an objection to the view that the gold was introduced subsequently to the formation of the banket-beds, that it occurs very regularly in certain beds, whereas others equally favourably situated in regard to the source from which the solutions are supposed to have originated, are barren. It is not easy to account for the erratic courses of solutions, especially if, as in Mr. Crosse's experiments, they are induced by an electric current.

A theory has been advanced in which the gold is described as having been precipitated from a solution, at the time of the formation of the banket beds. It is supposed that driftwood collected in some way by the pebbles, precipitated the gold around them.

An examination of the assay plans at the various mines, with an examination of the reefs, establishes, the author thinks, the following facts, which agree better with the theory that the gold was deposited after the formation of the banket-beds than at the time of their formation :—

1. That the gold is very evenly distributed in the south reef and its leaders, in the vicinity of Johannesburg.
2. That although the gold is pretty evenly distributed in the other reefs, yet certain richer patches or shoots occur, as in ordinary quartz-veins, and some poorer patches which are left in.
3. That exceptionally rich pockets are occasionally found in which the gold occurs along with pyrites.

4. That the richer patches or shoots are not necessarily in close proximity to igneous dykes, so far as these are at present disclosed, but, on the contrary, that the reefs are in some instances poorer nearer dykes.
5. That whenever, as sometimes occurs, the reef is richer than the average on one side of a dyke, it is poorer on the other side.
6. That frequently when the reefs are thinnest they are richest. Leaders, being thin, are usually much richer than the reef near which they lie.
7. In the south reef, the larger the pebbles, as a rule, the richer the reef. This applies to some extent also to other reefs, but there are exceptional cases in which the richer portions of the reef have smaller pebbles. Barren beds with large pebbles also occur in the banket formation.
8. At some mines the richer shoots have, like the prevailing faults, a north-westerly or south-easterly trend.

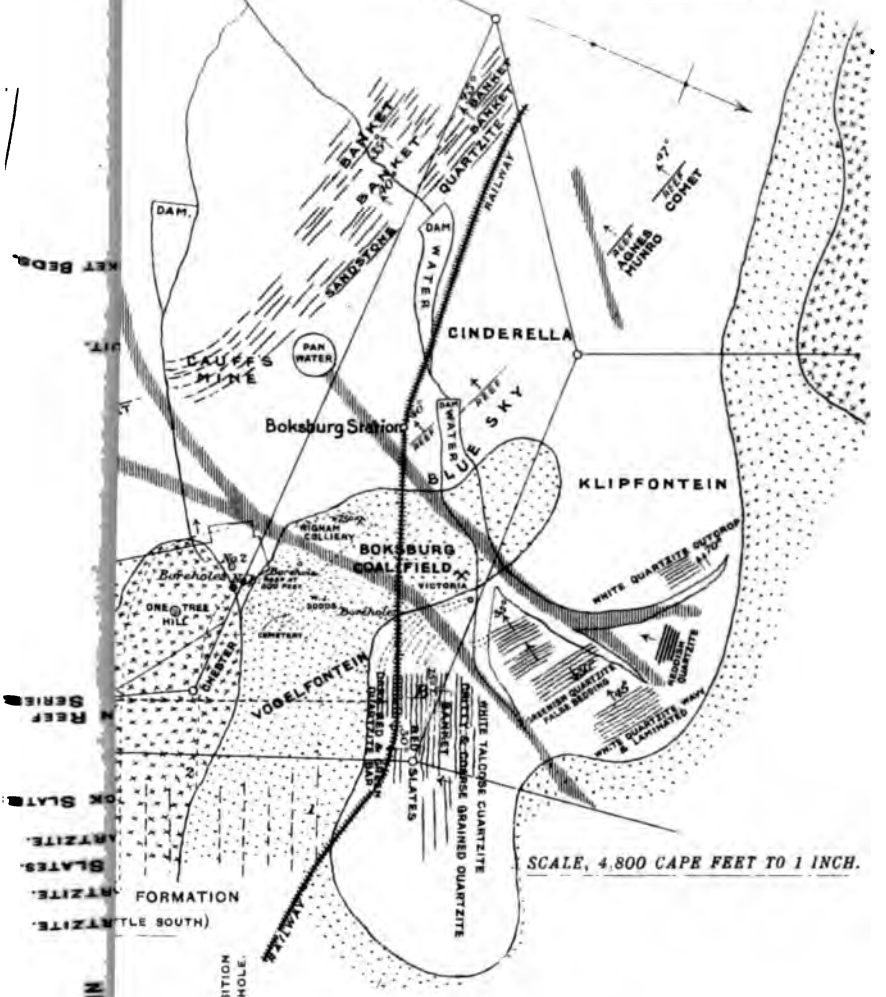
The geological plan and section (Figs. 2 and 3, Plate XX.) of the neighbourhood of Boksburg shows one of the still unsolved problems of the Witwatersrand. The reef worked at Knights, Balmoral, Driefontein Comet, and which is cut off at Blue Sky close to Boksburg station, has up to the present not been found eastward. Fig. 2 is an attempt by the writer to reason out the probable position of the reef preparatory to giving an opinion as to the value of certain claims with a view to starting operations on the property. The positions of the dykes are reasoned out from the disjointed outcrops of the banket formation. It will be seen that in the writer's opinion the banket formation turns sharply to the north-east on the eastern side of the dykes which he assumes occur. He believes that the formation has been slightly thrown up on the eastern side of the dykes, and that consequently the position of the Blue Sky Reef is most likely to correspond to that of No. 3 reef in the section (Fig. 3).

It will be noticed that the writer places the limit of the workable depth at 5,000 feet. He believes that it is possible to properly ventilate workings at that depth by properly opening out the mine. For this purpose the shafts should be round and of a large diameter. The rectangular shafts generally sunk on the Witwatersrand, besides being so timbered as to obstruct ventilation, are, as is well known, of an unsuitable section to obtain the maximum of ventilation. As shown by experiments made in 1884-1885 by the writer, and described by him in a paper entitled "The Temperature and Moisture of Air-currents in

FIG. 2 - GEOLOGICAL MAP

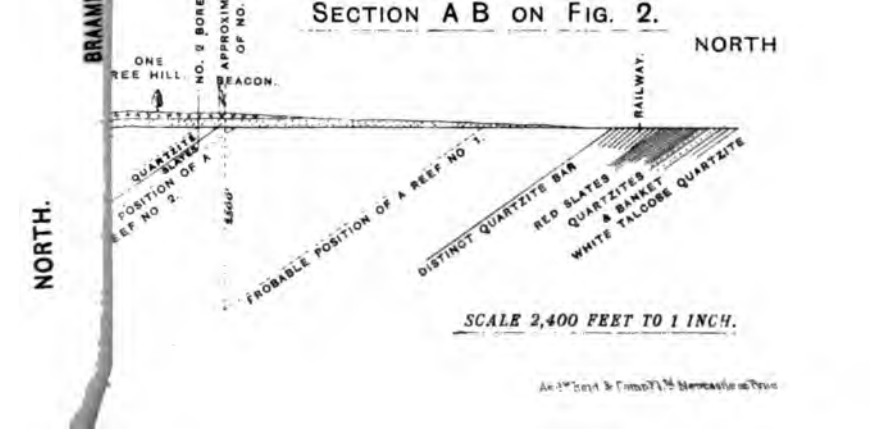
OF NEIGHBOURHOOD OF

BOKSBURG, WITWATERSRAND.



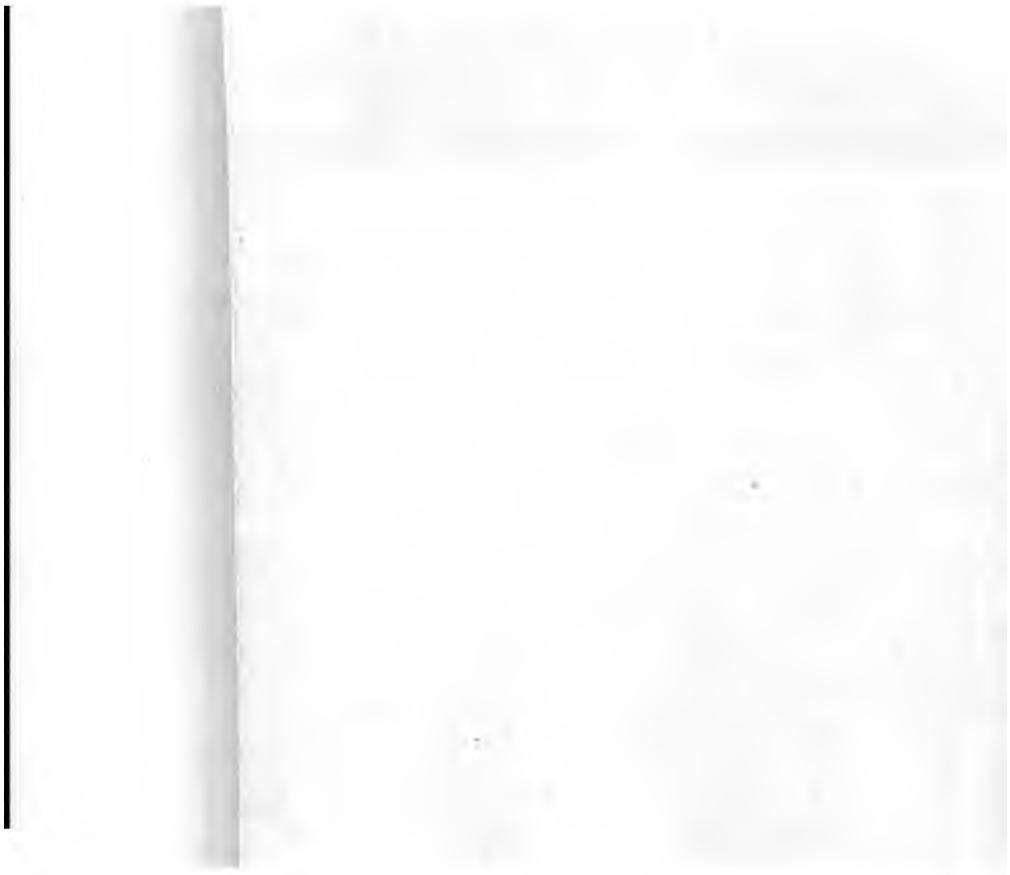
SCALE, 4,800 CAPE FEET TO 1 INCH.

SECTION A B ON FIG. 2.



SCALE 2,400 FEET TO 1 INCH.

...Bird & Cowell, Lith. Newcastle on Tyne.



Mines,"* 88 degs. Fahr. was the highest temperature recorded in the workings of the Dukinfield collieries, into the ventilation of which he was enquiring, at a vertical depth of 2,856 feet. There was no difficulty whatever in working at that depth and that temperature. Unless the Witwatersrand mines are laid out with a view to the effectual ventilation of large areas, such an oversight will cause considerable difficulty at greater depths than are being now worked, and the cause will be put put down to high subterranean temperatures.

Mr. JOHN M. LIDDELL (Newcastle-upon-Tyne) wrote that diorite dykes were, in his experience generally, if not always, to be found near gold and diamond deposits in South Africa, a fact which tended to show that there was something common at least to the agencies that produced both the diorite and the deposit of the precious metals and minerals. It had been suggested that the occurrence of two reefs near together, viz., the main reef and the south reef, was due to overlap or over-sliding of bed upon bed, but the great areas of the south reef and the main reef now proved to occur simultaneously show that theory to be untenable. The banket formation appeared to be rather an extensive system of strata containing beds of conglomerate of varying thickness, the whole series being faulted, tilted, denuded, or overlaid by subsequent action, many parts still remaining under cover, and the existence of gold in the conglomerate was discovered where the latter was tilted to the surface or denuded. It has been suggested by Mr. Alford that the gold-bearing beds were formed by the deposit of silica, gold, and other matters from a hot solution. This theory would explain the occurrence of the banket beds, and also of interbedded veins of quartz. The rich cross veins of quartz which were found running from main interbedded veins were observed, he (Mr. Liddell) thought, invariably to run into the foot-wall (once the floor), and never into the hanging wall (once the roof), of the vein, unless a portion of the vein had been tilted through an angle of more than 90 degs. Dykes, in his (Mr. Liddell's) experience, did not carry gold unless intersected by quartz veins. At small faults, there was frequently found to be some change in richness of the deposits, the banket on one side containing more, and on the other less gold. Probably the faults were formed just before or during the deposition of the banket (the fault-faces not being always clean cut), and thus formed natural riffles

* *Trans. North Staff. Inst.*

which retained more gold on one side than on the other. It is worthy of notice that the Carboniferous beds lie immediately upon the auriferous beds over a wide area. No case of fossils occurring in the banket beds was generally known, but it was nevertheless conceivable that the Carboniferous beds succeeded the auriferous beds in chronological order. The sandstone beds were very porous, possibly more so than the banket-beds which contained much clayey slime, and often have a mud-stone bed for a footwall. Deposits of coarse particles of gold and pyrites commonly occurred under pebbles in the banket, more especially where the pebbles were of good size, say that of a walnut, evidently drifted there by a water-current, just as sand was deposited behind a pebble by the ebb of a wave on the sea-beach. This suggested that the gold was deposited (perhaps precipitated) in a liquid on a pebbly bottom during the deposition of a gravel-bed moved by currents or waves. The pebbles always showed clearly in the deposit, often with metallic particles lying under them, and with layers of shaly matter about them. The south reef consisted commonly of about three thin banket beds with clay partings and silt between. It varied from a mere parting to a seam of as much as 15 feet in thickness, and where no pebbles were apparent the gold was often found in the sand-stone and clay partings. The reef (seam) was commonly richer where it was thinner, but the amount of gold from yard to yard was very irregular, although from a given area of say 20 acres, a fairly regular output could be maintained. The richness of thinner areas of reefs was probably due to the deposit of gold having been more or less constant, and that of the contemporaneous silt and gravel very variable in banks and ridges.

Mr. WALCOT GIBSON thought it was satisfactory to find that Mr. Sawyer recognized the existence of reversed faults and of considerable deformation of the beds in the Witwatersrand district. It had been usually considered that the beds lay in a simple synclinal, but it was evident from a consider-tion of the facts that this was only partially correct. The beds had certainly glided over each other. This made it quite within the bounds of rational conjecture that the gold-bearing banket-beds would be found nearer to the surface on the dip than would be the case if the beds lay in a normally-faulted synclinal. The practical bearing of the recognition of reversed faults and overthrusting was, as Mr. Sawyer stated, of great importance. It added to the value of all deep claims, and it was to be hoped that boreholes and trial-shafts would be sunk farther from the northern outcrop than it had been yet thought worth while to make any attempt with a prospect of success. The age of the banket formation still seemed to be uncertain. It was not safe in a folded district like

the Zwarteberg mountains to argue the age of beds from the presence of fossils, even in closely associated beds or beds in actual contact. The strata containing the fossils might only be a portion of another formation caught up and preserved in the fold. The data for ascertaining the relationship of the igneous dykes and intrusive masses to the gold-bearing conglomerates had not, as yet, been sufficiently carefully collected to lay down any positive rule that would be a guide to the miner. So far as at present known, we might safely look to the Rand yielding a steady and increasing output of gold, and to its forming an important factor in an estimation of the world's future output of gold.

Mr. R. T. Moore (Glasgow) wrote that he had read Mr. Sawyer's paper with great pleasure, and hoped that continued investigations of this nature would be made. The discovery of the Rand gold-field was so recent that there had been little time for fixing its position in the well-known geological divisions, or even for fixing the stratigraphical position of the various gold-bearing beds. One deep boring of 2,400 feet had been put down to the main reef at a point 4,500 feet distant from the northern outcrop, but a good stratigraphical chart had yet to be made. He (Mr. Moore) visited the district in 1894 and again in 1895. Both of these visits were in his professional capacity as a mining engineer, and time did not admit of his entering at any length into the theory of the formation. There was no doubt that the banket was a stratified deposit, and that in the Rand gold-field it was probably deposited over an area of at least 1,500 square miles, but whether the whole area contained the main reef or whether it was displaced over certain areas could only be proved by further investigations and mainly by boring. Mr. Sawyer's sections were very interesting, but he (Mr. Moore) was inclined to agree with the generally accepted belief that the beds formed a large basin, and, while dipping at the northern outcrop at inclinations up to 85 degs., gradually flattened to a point under the black reef, which was the centre of the basin. The Rand gold-field resembled very closely the Edinburgh coal-field. There, for a distance of 10 miles along the western outcrop, the seams of coal, called "edge seams," lie at a dip of from 70 degs. to 85 degs., and are worked at that inclination for 1,800 feet from the surface, and yet a mile back the upper seams of coal, which might correspond with the black reef, are found lying flat. The dip at the eastern outcrop is 1 in 5 and less. The main faults throughout the basin run at right-angles to the strike, as in the Rand gold-field, and there are no faults of any importance running parallel to the strike in the steep measures. Where the flattening had been got in the lower

seams, the dip changed from 70 degs. to 1 in 10 in a distance of 600 feet. Very probably the main reef series in the Rand might follow the same course and there might be a very large area of flat main reef lying at a much smaller depth than was now contemplated, possibly under 5,000 feet. This was a matter which could only be proved by boring; indeed before any of the deep reefs were attempted to be reached by sinking, there ought to be a good many bores put down. Few people at home had a proper idea of the immense extent of the mining operations on the Rand. All the workings yet made are on the northern outcrop, and mines are in operation on this line for a distance of over 40 miles. The output of ore during the month of March, 1895, was 261,000 tons, yielding 184,945 ounces of gold. These figures are obtainable from the Chamber of Mines, which publishes every month a return showing the amount of ore worked and gold got by each mine. The mines in many respects resemble coal-mines, over which, however, they possessed the great advantage that there was no difficulty in disposing of the output, the great object being to increase it as much as possible. There was thus no rivalry between different mines. A man did not object to another gold-mine being started alongside of his, because it might interfere with the sale of his products; he was quite willing to give any information to a new comer when once he had secured his own claims. The mines were fitted with first-rate machinery, no expense being spared to procure the best, and the workings were well conducted. With regard to shafts, a pair of shafts 15 or 20 feet in diameter could easily be made to pass 300,000 to 400,000 cubic feet of air per minute. Openings could be made between them, and a portion of the air sent through each opening, reserving as much for the workings as might be wanted. With large quantities of air, he had no doubt that the workings could be kept cool enough to allow of their being carried to a depth of at least 5,000 feet.

The PRESIDENT moved a vote of thanks to Mr. Sawyer for his interesting paper.

The motion was agreed to.

Prof. CLOWES read the following paper on "The Limiting Explosive Mixtures of Various Combustible Gases with Air":—

THE LIMITING EXPLOSIVE MIXTURES OF VARIOUS COMBUSTIBLE GASES WITH AIR.

By FRANK CLOWES, D.Sc., Professor of Chemistry in the University College, Nottingham.

The connexion of the author's name with the process of accurately testing for small quantities of inflammable gas in the air has led to frequent inquiries being made of him as to what proportion of an inflammable gas in the air will cause explosion when it is fired. He has thus been led to undertake experiments with the object of determining this proportion.

It is evident that there will be some one proportion of an inflammable gas in air which will lead to the most violent explosive effect when the mixture is kindled. But it is known that mixtures containing both larger and smaller proportions of the gas are also capable of being exploded by contact with a flame. The object of the series of experiments, the results of which are here described, was to determine the least proportion of the gas and the largest proportion of the gas, which, when mixed with air, are explosive at ordinary atmospheric pressure. These are termed the limiting explosive mixtures.

The mixtures were made in a glass cylinder of known capacity, 3 inches in diameter, and closed at one end. The jar was filled with water, and was then inverted with its open mouth in a pan of water. The measured volume of air was introduced into the cylinder, and the remainder of the water was then replaced by the combustible gas. A small xylonite ball was then passed up into the cylinder, the mouth of which was closed with a glass plate. After the closed cylinder had been removed from the water, the air and gas were thoroughly mingled by causing the ball to run up and down inside.

The cylinder was then removed into a dark room, and a small Bunsen atmospheric gas-flame was brought to its mouth. If a sheet of flame travelled quickly through the whole length of the cylinder, the mixture was considered to be combustible independently of the external atmosphere. Such a mixture, although it burnt silently in the small mass contained in the cylinder, would undoubtedly produce explosive effects if

it were kindled in larger quantity. If the mixture refused to kindle in contact with the flame, or if it simply burnt slowly from the mouth of the cylinder as the external air reached it, it was considered to be non-explosive at ordinary atmospheric pressure.

It was found that the mixtures were more readily kindled upward by a flame placed below them than downward by a flame placed above them. Accordingly, the composition of the limiting explosive mixtures varied with the mode of firing.

The experiments were carried out with several different combustible gases and with coal-gas. The limiting explosive mixtures varied in composition with the different gases employed.

In order to measure accurately small volumes of inflammable gas in making the mixtures, the gas was diluted with nine times its volume of air. In this way a tenfold volume could be dealt with. Allowance was made for the air thus introduced with the gas, in calculating the composition of the mixture to be experimented with.

The results obtained by repeated experiments made with the same gas were concordant. In every case the next percentage of gas, below or above as the case may be, failed to fire back.

The results obtained are tabulated below:—

LIMITING EXPLOSIVE MIXTURES OF VARIOUS GASES WITH AIR.

Combustible Gas Used.	Percentage of Gas mixed with Air.		Method of Kindling.
	Lower Explosive Limit.	Upper Explosive Limit.	
Methane or marsh gas	5	13	Upward.
	6	11	Downward.
Coal-gas (Nottingham)	6	29	Upward.
	9	22	Downward.
Water-gas	9	55	Upward.
Hydrogen	5	72	,,
Carbonic oxide	13	75	,,
Ethylene	4	22	,,

The conclusions drawn from the above results are as follows:—

1. When mixed with atmospheric air at ordinary atmospheric pressures, different combustible gases show different limiting explosive proportions.

2. The range between the lower and upper explosive mixtures is least in the case of methane or fire-damp. The range is widest in the case of hydrogen, but carbonic oxide shows an almost equally wide range. The limits in the case of water-gas are widely separated; with coal-gas the range of explosibility is less.

3. The tendency to explode is greater when the mixture is fired from below than when it is fired from above. Hence the lower limit mixture contains less gas, and the upper limit mixture contains more gas, when the mixture is fired below than when it is fired above.

4. Since the risk of explosion occurring, when a gas is mixed in unknown proportion with air, is diminished as the limits of explosibility approach one another, the gases which were employed in these experiments may be placed in the following order of increasing danger:—Marsh gas, ethylene, coal-gas, water-gas, carbonic oxide, hydrogen.

5. In every case the danger of explosion, resulting from a naked flame being brought into contact with a mixture of unknown composition, is greater when the flame is applied to the bottom of the mixture than when it is applied to the top.

6. It is interesting to note that Mr. Lewis T. Wright, when firing a mixture of coal-gas with air in a horizontal tube, obtained, as limiting explosive proportions of gas, 10·3 and 23 per cent. respectively. The lower and upper limits of horizontal explosion here correspond with those obtained by the writer for vertical downward explosion. They differ considerably from the limits for upward explosion. Hence it appears that horizontal kindling corresponds in its general character with downward kindling. It is not to be classed with upward kindling.

NOTE.—The coal-gas and the water-gas which were used in these experiments had the following percentage compositions by volume:—

COMPOSITION OF NOTTINGHAM COAL-GAS.

Olefine hydrocarbons	5·3
Carbon dioxide	0·0
Oxygen	0·2
Nitrogen	5·5
Hydrogen	48·2
Carbon monoxide	6·6
Methane, or marsh gas	34·2

COMPOSITION OF THE WATER-GAS.

Hydrogen	49·6
Carbon monoxide	40·8
Carbon dioxide	2·6
Nitrogen	7·0

Prof. CLOWES read the following paper on "The Change of Composition Produced in Air by Flames and by Respiration":—

THE CHANGE OF COMPOSITION PRODUCED IN AIR BY FLAMES AND BY RESPIRATION.

By FRANK CLOWES, D.Sc., Lond. Professor of Chemistry in the University College, Nottingham.

In a previous volume of the *Transactions* of The Federated Institution of Mining Engineers* the author stated the results of an investigation into the proportions of nitrogen and of carbonic acid which must be mixed with atmospheric air in order to enable it to just extinguish the flames furnished by various combustible solid, liquid, and gaseous substances.

It was shown (1) that carbonic acid uniformly exerted a more powerful extinctive effect than nitrogen did; (2) that the size of the flame did not produce any effect upon the amount of extinctive gas which must be introduced in order to cause extinction; (3) that whilst wick-fed flames were extinguished by an atmosphere of almost uniform composition, gas-fed flames required atmospheres of widely varying composition in order to produce their extinction.

The author has now made a series of experiments which are complementary to those published last year. He has allowed the various flames to burn in an enclosed volume of air until they were extinguished by their own products of combustion, and has then determined the composition of the atmosphere which caused the extinction of each flame. Since the pressure of the atmosphere around the flame was found to have considerable influence upon the extinction of the flame, the apparatus was arranged so as to maintain the enclosed air always at atmospheric pressure. Accordingly the extinction of the flames was dependent only upon the composition of the different atmospheres surrounding them, and was not affected by variations in their pressure.

The apparatus (Fig. 1) consisted of a glass bell-jar containing air. This jar was closed below by immersion in mercury, and was closed above by a cork carrying a U-shaped pressure-gauge and a tube with a stop-cock. The flame to be experimented upon was allowed to burn just above the surface of the mercury, by floating a suitable small vessel upon

* Vol. vii., page 419.

the surface of the mercury, or, if a gas was being burnt, the gas-jet was kindled just above the mercury. The bell-jar was then placed over the flame, the stopcock being open, and as soon as the bell-jar was closed by being immersed in the mercury, the stopcock was closed, and the air inside was maintained at atmospheric pressure by raising or lowering the bell-jar whilst watching the pressure-gauge. As soon as the flame was extinguished, and the air within the jar had cooled, a sample of this air was drawn off through the tap-tube into a Hempel apparatus, and was subjected to ordinary gas analysis.

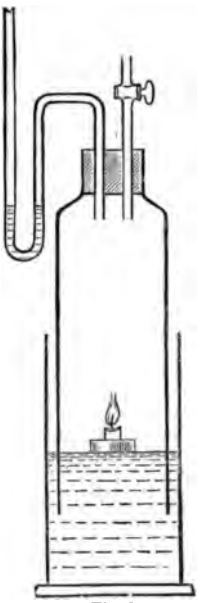


Fig. 1.

The results of the analyses are tabulated below, and are compared with the composition of the artificial mixtures of air with nitrogen which extinguish the same flames, and with the average composition of expired air. The samples of expired air which were analysed were drawn from the lungs themselves, and in some cases were expired immediately after having been inspired, whilst in other cases they were collected after the air had been in the lungs for forty seconds.

TABLE I.—COMPOSITION OF THE RESIDUAL ATMOSPHERES AND OF THE ARTIFICIAL ATMOSPHERES WHICH EXTINGUISH FLAMES, AND OF AIR EXPIRED FROM THE LUNGS.

Combustible Substances Burnt.	Percentage Composition of the Residual Atmosphere in which the Flame was Extinguished.			Proportions per cent. of O_2 and N_2 in which Flame is Extinguished when introduced.	
	O_2.	N_2.	CO_2.	O_2.	N_2.
I.					
Alcohol, absolute ...	14·9	80·7	4·35	16·6	83·4
„ methylated ...	15·6	80·25	4·15	17·2	82·8
Paraffin lamp-oil ...	16·6	80·4	3·0	16·2	83·8
Colza and paraffin ...	16·4	80·5	3·1	16·4	83·6
Candle	15·7	81·1	3·2	16·4	83·6
II.					
Hydrogen	5·5	94·5	—	6·3	93·7
Carbon monoxide ...	13·35	74·4	12·25	15·1	84·9
Methane	15·6	82·1	2·3	17·4	82·6
Ethylene (failed) ...	—	—	—	[13·2	86·8]
Coal-gas	11·35	83·75	4·9	11·3	88·7
III.					
Expired air (average)...	16·15	79·9	3·95	—	—
Fresh air	20·9	79·06	0·04	—	—

It will be seen from Table I. that the proportion of oxygen left in the residual air corresponds to that contained in the artificially produced atmosphere which had previously been found to extinguish each flame. Further, that the composition of the extinctive atmospheres left by the various common wick-fed flames was very similar, and closely corresponded with the composition of expired air.

A coal-gas flame, however, was able to reduce the proportion of oxygen in the air to a considerably greater extent than ordinary wick-fed flames; while the hydrogen flame diminished the oxygen in the air to about one-third the amount left by these flames, and to one-half that left by the coal-gas flame.

The combustion of fire-damp (marsh gas) produced an effect on the composition of the air which was very similar to that of the wick-fed flames.

The results obtained by Dr. Haldane on the respirability of atmospheres of various composition prove that the air in which either the above wick-fed flames, or the flame of fire-damp, have burnt until they become extinguished, are respirable not only with safety, but even without inconvenience. This is also true of air which has been once breathed, and which extinguishes ordinary wick-fed flames. Dr. Haldane further maintains that no permanent injury to health would result from breathing such atmospheres for some time. It follows that the extinction of the flame of a candle or of a safety-lamp in air does not prove such air to be unfit for respiration, and this accords with the experience of many practical miners.

It is worthy of note that the residual atmosphere which extinguishes a hydrogen-flame is irrespirable. The hydrogen-flame when burnt in air containing a diminished proportion of oxygen changes its colour from reddish purple to pale blue. This change of colour may serve as an indication of reduced proportion of oxygen in the air. The change of colour is probably due to the lowering of the temperature of the flame.

Dr. HALDANE said that Dr. Clowes' second paper had, of course, interested him very considerably because he had been working on the subject a good deal lately. It was instructive and interesting to find that all ordinary kinds of flame, with the exception of hydrogen or coal-gas, which contained hydrogen went out with about the same percentage of oxygen—the limits lying very close to one another. That appeared very clearly from Dr. Clowes' paper. He thought, however, that

Prof. Clowes' figures representing the percentages of oxygen which just extinguish flame were somewhat too low. In the experiments which the President and himself had published* they found that no candle or lamp could be kept alight after the oxygen was reduced to 17 per cent. He remembered one very careful experiment, in which they took two samples of air from the place where the candle or the lamp was actually being fed with air. They found, with the candle held vertically in the ordinary way, as in the case of a miner's lamp, that the light was extinguished at about 17½ per cent. of oxygen; they also found, by holding the candle horizontally, and doctoring the flame in every way, that they could get the candle to burn down to about 17 per cent. of oxygen, but it went out at 17 or 17·1 per cent. He had repeated these experiments in different ways, and always came to the same conclusion. He made one series in a laboratory, in which both the observer and the lights were contained in a large wooden airtight chamber. The results came out just the same as in a coal-mine. He thought that possibly the conditions in Prof. Clowes' experiments might not have been quite satisfactory for answering the question which was put before them. With the bell-jar, described by Prof. Clowes, it seemed to him quite possible that the air at the top of the bell-jar might become extinctive to flame, while the air at the bottom was still fairly good. In the experiments in the wooden air-chamber this point was clearly shown. If one took a candle and held it up to the roof of the chamber, it went out immediately, but when it was held at the bottom it burned perfectly well. The average composition of the air in the bell-jar might not represent the composition of what was actually extinctive to the flame. Prof. Clowes mentioned in the last paragraph: "It is worthy of note that the residual atmosphere which extinguishes a hydrogen-flame is irrespirable." Applying that to the conditions in coal-mines, it was certainly a very bad atmosphere which extinguished the hydrogen-flame, but nevertheless the President and himself had frequently gone into atmospheres which did extinguish the hydrogen-flame of Prof. Clowes' lamp. He would not say that they were not affected by this atmosphere. They both became more or less blue in the face, but still they did succeed in outliving the hydrogen-flame, and the oxygen percentages at which the flame was extinguished were considerably higher than that found by Prof. Clowes in laboratory experiments. When there was a little fire-damp, the flame seemed to be extinguished much sooner. The flame went out at an oxygen percentage which was still quite respirable and produced

* *Trans. Fed. Inst.*, vol. viii., page 558.

no very evident inconvenience or anything which would be noticed, beyond what might be attributed to the heat of the place. These results had considerable practical interest. One might fairly well outlive a hydrogen lamp if the flame showed a well-marked cap before it was extinguished. He thought he ought to mention those experiments, nevertheless he did not recommend any one to go into an atmosphere in which a hydrogen lamp would only just burn. It was certainly a dangerous atmosphere, and if they went a little farther, they would incur great risk of suddenly falling over and never rising again, especially if they fell down into an atmosphere which contained a larger percentage of black-damp.

Mr. A. L. STEAVENSON said that he would like to know Prof. Clowes' opinion on a point in connexion with the explosions: If they laid a train of gunpowder and set fire at one end it would run along the train without an explosion, because there was no resistance, but if they put that train where it could not expand they would have an explosion. He imagined that just the same conditions existed when they ignited a train of gas. No doubt when a flame of gas first ignited it travelled a long way as a simple flame, but when the expansion became so great that it experienced a resistance from the passages of the mine then it became an explosion. He would ask whether the tests which Prof. Clowes made in the open glass would not vary if the gases were contained in a cistern or vessel in which they could not expand? There was another matter which was not altogether to the point, which was this: It had often been remarked after an explosion in a mine that there was possibly a vacuum created, and that a second something, similar to an explosion, was caused, which was merely the rebound or the refilling of the passages by the air which had been forced out. He knew one case, where large baulks of timber, 15 or 16 inches square, had been driven outwards, and within 6 feet equally large baulks of timber, as large as his own body, were driven inwards, some being driven one way and some the other, showing opposite effects of the explosion. That had nothing to do perhaps with the limiting mixtures, but he called the attention of the Professor to that point to see if he could clear it up, as to whether it was not the effect of the rebound after the explosion. Turning to the table of limiting-mixtures, it seemed to him that Prof. Clowes rather differed from the results given by Prof. Lewes, who said that coal-gas was much more easily ignited than pit-gas or methane. From Prof. Clowes' remarks upon the results obtained by Dr. Haldane, he was led to think that Prof. Clowes was not aware that in 1864 and 1865 there was a Royal Commission on metallic mine

That Commission visited Cornwall, where at one time the mines were in a very bad state; they also visited Cleveland, where, for want of mechanical ventilators, the air in some of the mines was not what it was to-day, and on that occasion they not only took samples of the air and had them tested, but had very valuable experiments made and the results published in an Appendix. There was a large amount of valuable information on that subject in the report of that Royal Commission. There was one other subject which he believed had never been dealt with, and that was the effect of carbonic oxide on human life, and the point at which carbonic oxide became dangerous. He had looked into that question, but if the professor on this or any other occasion could give them a little hint as to the limits up to which carbonic oxide might be present in their mines, he thought that it would be useful. Their attention had been brought to it by the use of gas-engines and petroleum-engines in mines.

The CHAIRMAN said he believed that Dr. Haldane had recently been doing something with carbonic oxide, although he was not sure that he was in a position, or perhaps wished, to give the results of his investigations.

Dr. HALDANE said that he would be happy to give any results which he had obtained which might be of interest to the Institution. A paper by him on the action of carbonic oxide would appear in the next number of the *Journal of Physiology*,* dealing with the action of that gas. As regarded the poisonous percentages of carbonic oxide for animals, there had been a good many experiments already made, some of which he had repeated. The smallest percentage of carbonic oxide which visibly affected such small animals as a mouse was 0·06 per cent.—that was to say, roughly speaking, one part in 2,000. What the smallest percentage was which would affect a man it was not easy for him to say just now, but he believed that it was about the same percentage, 0·06. He had been making a good many experiments with carbonic oxide, and had made some upon himself lately to try to make out what the limiting-percentages were which produced distinct symptoms, and he found it very difficult. It took a long time for very small percentages of carbonic oxide to act upon a man, but he could certainly guarantee the fact that 0·2 produced very awkward symptoms. After breathing it for an hour or so he began to stagger when he walked about. He felt in the same condition as a man who had been in after-damp or gob-stink. In these experiments he had examined his own blood, and found that carbonic oxide was present in the blood long before it produced any outward symptoms. It gradually accumulated

* Vol. xviii., No. 3.

in the blood until finally it reached the point where the blood was so saturated with carbonic oxide that it would no longer convey sufficient oxygen. With regard to a practical test for carbonic oxide, there was great difficulty in detecting it in a mine except by the symptoms. A lamp might burn in an atmosphere in which after-damp and carbonic oxide were present in sufficient quantity to knock a man over in a short time. The test that he should propose for carbonic oxide, and which he had shown by experiment to be a good test, was simply that of taking a mouse along with one into these atmospheres in a little glass cylinder, or anything open to the air; the mouse would be visibly affected by the poisonous action of the carbonic oxide in the atmosphere in two or three minutes, or less, whereas to produce the same effect upon a man it might take ½ hour or more. By taking in a mouse they could be perfectly sure that they would not be poisoned without due warning of the danger. He had tried this himself, and the mouse was rendered powerless in two minutes, and he continued breathing the atmosphere for at least ½ hour and was not at all affected. In smoke or gob-stink, or gunpowder smoke or after-damp, this test might be of some use, and possibly there might be some present who might find it useful some day to put it into practical application.

Mr. Lewis T. Wright (London) remarked that in his experiments the tube was placed in a horizontal position for convenience in operating, but any other position would have given the same results. The difference in the upward and downward modes of firing, which had no bearing upon the method employed by himself, however, appeared to have an important influence upon the limits in the case of the mode of operating adopted by his friend Dr. Clowes. In this case the column of explosive mixture was stationary, and at the moment of removing the cover of the cylinder in order to introduce the igniting-flame to the mixture it was very possible that some slight alteration in the composition of the layer of explosive mixture in contact with the igniting flame might take place. Such alteration undoubtedly occurred, though it was also likely that with such skill as Dr. Clowes would bring to bear upon the experiments there was but a trifling alteration. This alteration in the composition of the prepared explosive mixture was more likely to be felt in the case of downward firing than with upward firing, because when the mouth of the cylinder was upward the air would more readily enter from the outside than when the mouth of the cylinder was downward. In this latter case the lighter mixture in the cylinder would tend to remain in the cylinder, and only suffer alteration in composition by slow diffusion with the outer

air. In the case of the cylinder being mouth upwards, which position answered to downward firing, the heavier air would tend by gravitation to rush into the cylinder and displace the lighter mixture. The position of the cylinder was also important as regards the action of the igniting flame. The bunsen burner would be held more or less in a horizontal position, in order to project its flame into the mouth of the cylinder. The flame would bend itself upwards, so that when the cylinder was held downwards the flame readily entered into the gaseous mixture it was intended to fire. On the contrary, when the mouth of the cylinder was upwards, which was the position answering to downward firing, the bunsen igniting-flame was bending away from the explosive column. Sir H. Davy pointed out that the mass of the igniting-flame or body exercised an influence, and they could understand that a mixture that might not be inflamed by a small focus of heat might be ignited by a larger focus, even though the grade of temperature of the igniting focus of heat should be no higher. The larger focus would heat up the explosive mixture, and it was well known that by pre-heating a mixture otherwise non-explosive it might be made to explode. Under all circumstances it would seem that upward firing was the method of the two employed by Dr. Clowes most free from liability to disturbance, as also most sensitive by the possibility of better approach of the igniting-flame to the mixture to be ignited. He had found 5 and 13 per cent. for the lower and upper limits of marsh-gas. It was worthy of note that Messrs. Mallard and Le Chatelier found, by operating with tubes in which the position was not of significance, 5 and 16·8 per cent. The lower limit being identical with that found by Dr. Clowes with upward firing—the method that would appear to be the most accurate. The knowledge of these limits was most important. If, for instance, an air-mixture of 5 per cent. of ordinary coal-gas or marsh-gas was explosive, it was evident that they must somewhat increase their suspicion of small admixtures of such gases with air. In his (Mr. Wright's) experiments, conducted with tubes of $\frac{3}{4}$ inch diameter, an air mixture containing 9 per cent. coal-gas would not light back into the tube. The wide limits of water-gas, due no doubt to the hydrogen and carbonic acid that it contained, rendered it distinctly more dangerous than coal-gas. In the course of his experiments he (Mr. Wright) had found that the limits of explosibility as determined by his method could be extended about 2 per cent. either way by previously heating the mixture, and, further, he had found that mixtures in which in an ordinary way a flame would travel but slowly could be made to develop an explosion of great severity under the influence

of the vibration of its own flame. If the flame commenced vibrating rapidly, it often happened that suddenly the mixture developed a detonating character and became violently explosive. Thus a mixture that was at the limits of explosibility might, under the influence of the vibration of its own flame, be really a very explosive one. It was well-known that such explosions could occur in four stages. Firstly, an uniform and regular passage of the flame; secondly, the vibration period; thirdly, a violently explosive condition, followed sometimes by, fourthly, the extinction of the flame. He thought that he had been previously able to explain the reason of the development of the third or highly explosive stage,* and he had lately obtained some evidence that explained the reason of the vibratory period—which was due to the flame becoming alternately mixed with and freed from the products of its combustion.

Mr. JAMES ASHWORTH (Derby) wrote that he had often called attention to the great difference there was in point of safety between a gunpowder of the very best class, and that which is in general use. The former was calculated to produce a large volume of carbonic acid gas, and the latter a still larger volume of carbonic oxide, and there were instances recorded where this gas had been purposely lighted by the collier after a shot has been fired. The danger of this fact had not been recognized in the past, and it was still an unrecognized fact. The first intimation of its recognition was contained in the report of Mr. Stokes, H.M. Inspector of Mines for the Midland district, where he said "the quality of an explosive is a material factor with respect to its safety, gunpowder when used should be of an equal composition, and not contain less than a stated percentage of saltpetre, thoroughly incorporated, and of a certain sized grain. The high explosives should also be of an uniform mixture enclosed in a preservative description of cartridge, so that a charge of cartridges may be detonated when the shot is fired, and no portion allowed to remain unconsumed either at the end of the shot-hole or amongst the mineral blown down by the shot of which it formed a part." The idea contained in Mr. Stokes' remarks being the production, if possible, of perfect combustion and therefore of carbonic acid gas. Prof. Clowes stated that the ignition-point of carbonic oxide extends over a very long range, viz., from 13 to 75 per cent., whereas marsh-gas has a range of from 5 to 13 per cent. only. It appeared from Prof. Clowes' experiments that if any explosive on ignition or on detonation produced more than 13 and less than 75 per

* *Proceedings of the Royal Society*, vol. xlii.

cent. of carbonic oxide in the air, we could justly look on it with more or less suspicion as containing a certain element of danger. Explosives which did not produce carbonic oxide on explosion or detonation have in like manner a very important point in their favour. Prof. Clowes' experiments were made at ordinary atmospheric pressure, and therefore we cannot tell how much this wide range of danger from explosives producing carbonic oxide might be increased when the gases were under the great pressure of explosion and when confined in the shot-hole. It was, however, very evident that if the percentage of carbonic oxide exceeded 75 per cent., it became an extinctive gas until the percentage was reduced by the addition of more oxygen. This fact afforded an explanation of certain phenomena which he (Mr. Ashworth) had noticed in connexion with the effects of blown-out shots and the resulting disasters, viz., that the products of the explosion of the blasting-powder or other explosive had been projected into the atmosphere in such a state, that as soon as additional oxygen was supplied, another and entirely independent explosion was initiated, and its occurrence was distinctly indicated by the space in the pit-level where no damage was done, and where all the surfaces were under equal pressure. This space he had previously named the "incubatory space," and following the present argument, he thought it was a very applicable one. A further fact supported the carbonic oxide theory, viz., that all the people found within the incubatory space had been almost instantaneously killed by the poisonous atmosphere before the second explosion could take place. Supposing that this argument were correct, he considered that the late explosion at Timsbury colliery, and also that at Malago Vale colliery, were fully and amply accounted for without considering any part played by coal-dust, and that the coal-dust was coked by the heat set free when the carbonic oxide was burnt into carbonic acid. It would be extremely interesting if, at some future meeting, Prof. Clowes would give an additional table showing the lower explosive limit of mixtures of methane and carbonic oxide, together with the same limit for carbonic oxide when from 1 to 5 per cent. of steam was added. If Prof. Clowes' paper on "The Change of Composition produced in Air by Flames and by Respiration" were considered in conjunction with his first paper, much light would be thrown on the after-damp produced in the immediate neighbourhood of a blown-out shot, and one might thus account for the evidently very sudden, in fact, practically instantaneous death of everybody within say 300 feet of an explosion. Mining blasting-powder produced from 30 to 50 per cent. of its resultant gases as carbonic

oxide. Prof. Lewes gives 33·75 per cent. as the proportion, and therefore if this volume of carbonic oxide were projected into a mine roadway and its combustion completed, the residual atmosphere would be (according to Prof. Clowes' Table II.), oxygen, 13·35; nitrogen, 74·4; carbon acid gas, 12·25; but as this second explosion also included the combustion of the floating coal-dust, and of that disturbed by the violence of the blown-out products of the gunpowder, we have again a vast flame of carbonic oxide pushed forward by the second or incubatory explosion, and acting on the whole atmosphere of the mine, which would thus be enormously compressed. This compression was estimated by Mr. A. L. Steavenson as equal to 200 pounds per square inch.* Under these fresh conditions it did not seem improbable that the nature of the explosion was entirely changed, and that from this point the explosion became one of the oxygen of the air (in the presence of the floating coal-dust), of the nitrogen of the air, and of the watery vapour present in the air, instead of as usually understood, the combustion of coal-dust in the presence of the oxygen of the air. Prof. Berthelot gives the possible speed of the explosion of carbonic oxide and oxygen as 1,940 metres per second, and Prof. Dixon speaks of the detonation of these two gases in the presence of from 5 to 6 per cent. of steam in a long tube; it seemed possible, therefore, that the velocity might reach 20,000 feet per second when detonated.† Prof. Dixon also states in his paper on "The Rate of Explosions in Gases" that carbonic oxide will not burn in the presence of dry air; it would, therefore, further add to the interest of Prof. Clowes' paper if he gave the number of grains of moisture present in the air at the time when he made his experiments.‡ In conclusion, he (Mr. Ashworth) thought that Prof. Clowes' paper contained a definite warning, when considered from the coal-dust point of view, viz., that explosives which produce carbonic oxide on explosion or on detonation should only be used with very great discretion in coal-mines.

Mr. JEREMIAH HEAD (London) said he thought that mining engineers, and engineers generally, might be very thankful that there were so many purely scientific men who were at present engaged in experiments on that most important question. At present it seemed to him that purely scientific questions were very far from being fully exhausted, but they were getting guidance day by day from the labours of these gentlemen. He took it that what had been in their minds to a great extent had been to see how far gases which were sufficiently denuded of oxygen

* *Trans. Fed. Inst.*, vol. iii., page 321.

† *Ibid.*, page 317. ‡ *Ibid.*, page 316.

to prevent flames of different kinds from existing were incompatible with the existence of human life. It did not seem to him, however, that the things could be quite compared. In the case of burning combustibles in a mixture of air and oxygen and other gases, it was mainly a question of the proportion of oxygen present, but in supporting human life it was not so. Then, as had been mentioned by Prof. Haldane, carbonic oxide acted in the way of poisoning whilst carbonic acid acted in the way of drowning. They knew that some three or four years ago, at the Leeds Forge, two or three men were sitting in a cabin in which there was a slight leakage of water-gas (containing carbonic oxide). These men were poisoned in the same slow way as that mentioned by Prof. Haldane, perfectly unconsciously, and eventually died. Prof. Haldane had told them about lowering a flame into a well with a man, and he had told them that the flame went out when 17 per cent. of oxygen was present; he had not told them what the percentage was when the man died, perhaps he would tell them that later. Prof. Huxley, in his work on *Elementary Physiology*, said that the air expired from their lungs was 5 per cent. poorer in oxygen and 5 per cent. richer in carbonic acid than the air which they inspired. It appeared from Prof. Clowes that the total quantity of oxygen in the air was about 21 per cent.; and the expired air would contain 16 per cent. of oxygen. He presumed if they were confined in a room where there was nothing but air that had been expired from human lungs, they could not live, although there would still be something like 16 per cent. of oxygen. It appeared that the men could breathe the same air over and over again; after once breathing it contained 16 per cent. of oxygen, and if life could be sustained with 7 per cent. it was clear that the vitiated air might be breathed about three times over and still maintain life.

Prof. CLOWES, in reply to Dr. Haldane's remarks that he found 17 per cent. of oxygen the lowest proportion that would support a flame, said that if he looked to his early work he would find that that was the proportion which he had himself obtained; his results varied between 16·2 and 17·2, thus coming practically to the proportion at which Dr. Haldane had arrived. The different percentages which he had obtained in the later set of experiments was mainly due to the fact that the air was confined for sometime around the flame and became strongly heated, and the higher temperature helped to maintain the flame, even when a lower percentage only of oxygen was present. Dr. Haldane thought that a bell-jar vessel was not suitable as compared with a broader vessel, such as a room or a cubical vessel, but the illus-

tration of the vessel in the paper was not drawn to scale. The jar was really much broader and lower; they found the mixture of the atmosphere inside throughout was very satisfactory, and he thought the explanation which he had already given was quite sufficient to explain the discrepancy to which Dr. Haldane had referred. Dr. Haldane also referred to the fact that the hydrogen-flame was extinguished with larger proportions of oxygen than he had mentioned in the paper. He believed on those occasions he was working with air in a mine which contained a proportion of carbonic oxide and certainly of fire-damp. He had himself discovered the fact that the presence of fire-damp tended to extinguish the flame. The conditions in Dr. Haldane's experiment were different from his own, since extinctive inflammable gas was present to aid the action of the reduced proportion of oxygen. He was very careful in his paper to mention the conditions under which the tests were made, so that anyone might satisfy himself as to the way in which the results were arrived at. Mr. Steavenson had suggested that the mixtures which he could fire without explosion might be explosive if they were confined. That was very probably the case; if they were large in quantity and in a confined space they would certainly explode. Mr. Steavenson was, however, mistaken as to Prof. Lewes' remarks. Prof. Lewes referred to the temperature at which coal-gas could be kindled as compared with marsh gas or fire-damp, and said that coal-gas was kindled at a much lower temperature than fire-damp; he did not refer to the composition of the mixtures which were explosive, but to the point of ignition only, so that Prof. Lewes' statement did not in any way militate against his own. He was much interested in the statements made by Dr. Haldane as to the poisonous proportion of carbonic oxide, and though an American observer had given different figures, he thought that Dr. Haldane's were more deserving of their trust. Mr. Wright suspected a change of composition in the cylinders when the cylinders were opened. He had taken every precaution against that, and he did not think it could have occurred to any noteworthy extent. The flame was held over the cylinder before the cover was withdrawn. The kindling was instantaneously effected and was watched in a dark room, so that if any change of composition occurred it must have been an extraordinarily rapid process. With regard to the inverted cylinder, he did not think such a change was possible, as he had kept it inverted for several minutes, and had found the mixture within was still unchanged in composition. In both cases a stiff and straight flame was rapidly thrust into the mixture. However, he was going to

repeat the experiments by means of closed vessels and ignition by electric sparks, so that then there would be no possibility of change of composition. One fact stated by Mr. Wright had interested him very much, and it was important in that connexion, viz., that temperature had a very great effect on the explosibility of the mixture. That was an important fact, and showed how an explosion might be extended in a coal-mine; it might arise in an explosive mixture and extend into non-explosive mixtures if it got into the hotter parts of the workings. It was also the fact that a non-explosive mixture which could not be kindled and exploded might be kindled by some amount of vibration. Mr. Head's remark as to the difference between the respirability of the atmosphere and its suitability for maintaining combustion of flames was a matter that he had thought of before. In fact, he was constantly being asked by his mining friends whether he could not give them some test, not only for inflammable gas, but for other gases which were noxious to breathe. He was glad that Dr. Haldane was on the track of carbonic oxide, which was most poisonous; and if the mouse test proved to be a reliable one they must congratulate him upon suggesting it, for they had no test at present beyond those of the chemical laboratory. He trusted that the present so-called Anti-vivisection Act would not prevent the use of a living mouse for so important a purpose. Mr. Head was correct in supposing that his paper referred to expired air which had been drawn once into the lungs and once expired, and not that which had been drawn in over and over again. The same air might be breathed in again and again three or four times before it really became inconvenient and uncomfortable to breathe. The proportion of moisture present in the mixture of carbonic oxide and air was more than sufficient to insure its kindling. The mixture used was never dried as in Prof. Dixon's experiments referred to by Mr. Ashworth.

Mr. J. A. LONGDEN (Stanton-by-Dale) proposed a vote of thanks to Prof. Clowes for his very valuable papers.

Mr. E. B. WAIN seconded the proposal, which was very heartily adopted.

The following paper by Dr. L. van Werveke on "The Occurrence, Mode of Working, and Origin of Petroleum in Lower Alsace" was read:—

THE OCCURRENCE, MODE OF WORKING, AND ORIGIN OF PETROLEUM IN LOWER ALSACE.

By Dr. L. VAN WERVEKE.*

1.—Geological Structure of the District.

The Vosges, the mountains of the Haardt, the Black Forest, and the Odenwald form two anticlinals, which strike right across the Rhine valley, and are separated by a synclinal, the central line of which runs from Pfalzburg to the Kraichgau district. Another synclinal separates the southern anticlinal from the Jura. This southern anticlinal embraces the Vosges and the Black Forest, while the northern and less steep anticlinal includes the Haardt, Odenwald, and Spessart. The former is largely made up of pre-Triassic rocks, and the ancient core of the Vosges massif had been built up ere the Bunter Sandstone deposits began to mantle over its sides. In the northern anticlinal, on the other hand, the pre-Triassic rocks play a less predominant part, and there the overlying Bunter Sandstone forms a more continuous sheet: this anticlinal, together with the synclinals of Pfalzburg and Saargemünd rises north-eastward. The uplift took place presumably in Cretaceous times, and there is evidence to show that Jurassic strata, now long vanished, then overlay the Trias.

In this area the Cretaceous period was not one of deposition but of continental conditions. Tertiary strata, on the other hand, are extensively developed. The oldest of these are freshwater limestones with underlying pisolitic iron ores and brown coals in Lower Alsace. These strata are probably of Middle Eocene age (they have yielded mollusca and vertebrata), and they are overlain by unfossiliferous coarse conglomerates. The materials of the conglomerates are exclusively of Jurassic, mainly Oolitic, origin, a fact which not only serves to confirm the hypothesis that Oolitic strata at one time stretched away over the whole area, but it also proves that although this area was not then exactly a level plain, neither was it one of a mountainous character: the earth-movements which at a later period resulted in the depression of the Rhine Valley had not then begun. The conglomerates are either Upper Eocene or very low down in the Oligocene.

* Translated by L. L. Belinfante, B.Sc.

Above these come Lower Oligocene marls, from which the crude oil is got in Lower Alsace: they are very thick, and are overlain by bituminous limestone, which is in its turn succeeded by the Middle Oligocene Septaria-clay. The marls are grey or greyish-green, very rarely red or chocolate-brown in colour, and sandstones and sands are interbedded with them. Of minor importance are certain calcareous inclusions, gypsum, anhydrite, and iron pyrites. These strata have been well determined from two deep borings made in recent years, one on the northern fringe of the Hagenau forest (2,034 feet), and the other at Oberkutzhausen (1,673 feet). Certain red marls were struck here among the Tertiaries which, in all but their vastly greater thickness, bore a wonderful resemblance to the red marls of the Middle Keuper.

The sands form irregular masses, lenticular in cross-section, of a maximum length of 2,600 feet, maximum breadth of 197 feet, and maximum thickness of 20 feet. Their mean strike is N. 22 degs. E., and they dip Rhinewards. At one time the petroleum was got in these sands by mining. They are surrounded by a zone of dark, bituminous, lignitiferous marls, which the mining folk used as an indicator. These marls are very rich in molluscan remains, and they show that the oil-bearing strata are of brackish-water origin.

The real thickness of the Lower Oligocene marls is unknown; at all events it has been proved to exceed 1,970 feet. Near Schwabweiler, the Lower pass into the Middle Oligocene marls (with a fauna largely made up of foraminifera) and these are overlain by grey sandstones with the remains of a tropical flora, the chief representative of which is *Cinnamomum polymorphum*.

A point worth noting is that the same stratigraphical relations obtain near Altkirch, in Upper Alsace, showing that the oil-bearing strata of that district belong to the same geological horizon as those of Lower Alsace. The significance of this fact will be better understood when we come to speak of the origin of the petroleum.

The bituminous limestone previously mentioned comes on over the Lower Oligocene marls at Lobsann. It is interbedded with thin layers of brown coal, and is seen to thicken the nearer one gets to the mountains. Besides rock impregnated with 18 per cent. of bitumen, there occurs pure limestone; this, however, often surrounds "veins" and patches of the bituminous limestone. The bitumen cannot be got out of the limestone by boiling, as it can out of the bituminous sands of Lower Oligocene age.

The brown-coal layers, which vary in thickness from a fraction of an inch to 18 inches or so, contain numerous plant-remains, small globular

masses of amber, land and freshwater mollusca, and ungulata, such as *Anthracotherium alsaticum*, *Entelodon*, and *Hyopotamus*. The brown-coals contain much pyrites: with them and the limestones conglomerates are here and there intercalated, increasing in thickness both north-east and south-west. These are made up of materials derived from the Muschelkalk and the Bunter Sandstone, and are therefore to be distinguished from the previously-mentioned coarse conglomerates of Jurassic (Oolitic) material, typically developed at the Bastberg. Vast denudation must have taken place in the long ages which elapsed between the deposition of these two conglomerates, and the Lower Oligocene marls represent this period—during which Lias and Keuper were being denuded away in Lower Alsace. This explains why these marls are marls, and why they so closely resemble the Keuper that in the mud brought up by the borings it is extremely difficult to say what is Tertiary and what is Trias. And yet the ability to settle that point is of considerable economic importance. The second conglomerate is proved by palæontological evidence to be of Middle Oligocene age, and is the equivalent of the marine sands of the Mayence basin.

The overlying Septaria-clay is also of marine origin, and is crowded with remains of foraminifera.

2.—The Depression of the Rhine Valley.

The movements which brought about this depression, which represents a difference of level of close upon 2,300 feet, evidently took place after the deposition of the Middle Oligocene. A glance at the geological map of Alsace-Lorraine shows that the sagging movement split up the Mesozoic rocks into three faulted areas, those of Zabern (Saverne), Rappoltsweiler (Ribeauvillé), and Gebweiler: just as these are cut off from the mountain-ranges by a series of faults, so are the Tertiary and alluvial areas cut off from them by a great fault. Innumerable minor faults traverse the Mesozoic areas. In the northern portion of the Zabern district, to which our attention must be mainly directed, these minor faults almost invariably strike N. 53 degs. E. In the southern portion, the stratigraphical relations are more complicated, owing to the presence of another series of faults striking due N. and S.

Mining operations have shown that the Tertiary rocks also are traversed by faults, but for various reasons these cannot be mapped out accurately; in the main, they probably run parallel to the great boundary-fault. In a word, it does not appear that any faults of tectonic importance strike from the Vosges towards the Rhine. The principal fault in its

course from Merzweiler to Weissenburg has a mean strike of N. 45 degs. E. The petroleum district of Schwabweiler has been subjected [by it] to greater dislocation than that of Pechelbronn.

Now we come to consider the direction of downthrow. In contradiction with the views of Messrs. Élie de Beaumont, Daubrée, Laspeyres, and Schumacher, Mr. Andreae has recently asserted that the great Rhine Valley fault dips down towards the mountains. But the writer could find no adequate evidence for this interpretation, and he adhered to the older and generally accepted view of the downthrow being towards the Rhine. If Mr. Andreae's theory were correct, the engineers who wish to bore for petroleum along the great fault itself could hope to strike oil only when they put the borings down in Mesozoic strata, at the points where these are brought against the Tertiaries; but, according to the other view, the borings may be put down through the Tertiary strata themselves.

The series of faulting movements was followed by volcanic eruptions, of which the Kaiserstuhl in Southern Baden is perhaps the most imposing relic.

3.—The Winning of the Petroleum.

The bituminous spring of Pechelbronn, to which that village owes its name, has been known for many centuries, and indeed the use of petroleum was already termed "ancient" in 1498 by Wimpfeling. The first underground workings in the oil-bearing sands were begun in 1742, and in 1768 these workings came into the exclusive possession of the Le Bel family, in whose hands they remained until 1888. Mining had gone on to a depth of 296 feet below the surface when operations were stopped at Pechelbronn at the end of that year. The method was to put down shafts to the oil-bearing layers of sand and work along these; but when great difficulties arose through outbursts of oil and gas, and the inrush of water, parallel levels were driven in the "hanging wall." Ten larger deposits and a great number of smaller ones, worked from eleven different shafts, represent the sum total of the old mining industry of Pechelbronn. Besides the oil which oozed naturally from the sand, the sands themselves were collected, and about 4 per cent. of crude oil was extracted from them by distillation.

Boring operations up to the year 1880 were only carried out to determine the direction of the trial-levels, which had to be driven above the oil-deposit. But since that year they have assumed continuously-growing importance, and now in the Pechelbronn oil-field alone more than 500 boreholes have been put down. Mining-Councillor Jasper*

* *Das Vorkommen von Erdöl im Unter Elsass*, Strasburg, 1890.

has given full details of the results of these borings. The oil-deposits which have been thereby proved occur at five different horizons, at the respective depths of 262-295 feet, 394-492 feet, 590-656 feet, 754 feet, and 1,098 feet.

The results of the borings vary considerably; about 23, or 4 per cent. of the whole number have struck natural wells wherein the gaseous pressure is sufficient to spurt the oil out at the surface with considerable force. One great advantage of the Lower Alsace oil-wells is that the oil in them is mixed with comparatively little water.

At Sulz and Wald petroleum was got in the eighteenth century, just as at Pechelbronn, but the mining operations have been long suspended. At Schwabweiler ($3\frac{3}{4}$ miles south-east of Pechelbronn) search was first made for petroleum about 1830, and a mining concession was granted by the King of the French in 1841; workings were carried on till 1883 by means of shafts reaching to a depth of 230 feet. Fifty trial-borings put down in the neighbourhood of the workings at depths varying between 260 and 295 feet had given unsatisfactory results. The petroleum-bearing sands of Schwabweiler do not not form long narrow bands, as at Pechelbronn, but spread over large areas in deposits having a maximum thickness of 6 or 7 feet. The dip of the beds is steeper, and small faults are numerous.

Simultaneously with the greater extension of the boring operations at Pechelbronn, the exploration of the areas lying outside the then known oil-field was pushed forward, the greatest activity in this direction being shown in the year 1890 when, as a result of the experimental borings, no less than 346 leases were granted. The occurrence of petroleum in Lower Alsace was proved in districts extending far beyond the Schwabweiler-Pechelbronn area, and not only in Tertiary rocks but also in the Mesozoic strata of the Zabern fault-area. In 1892, only two further leases were applied for, and in the annual report of the district governor of Lower Alsace for 1893 it is stated that the proposed heavy increases in taxation have deterred applicants for further petroleum-leases.

The average depth at which the borings struck oil in the Tertiary area was $256\frac{1}{2}$ feet. Curiously enough, south of the Zorn, where the actual average is only $134\frac{1}{2}$ feet, the oil-bearing Lower Oligocene strata occur actually at a greater depth from the surface than at Pechelbronn (where oil is not generally struck till about 294 feet). Between the Zorn and the Moder, and east of the Tertiary dislocation, the older Tertiary is almost everywhere covered by Pliocene and Drift; where it does crop out in places it is seen to belong to the Middle Oligocene. For these and other

stratigraphical reasons one infers that south of the Zorn it is not the equivalent of the Pechelbronn oil-bearing strata which has been struck by the boreholes, but an horizon higher up in the series, probably Middle or Upper Oligocene.

The oil is generally of a dark brown colour, and in a few cases light yellow, light brown, or dark green; it varies much in fluidity, and in places assumes almost the viscosity of asphalt. It has been worked since 1880 at Biblisheim, Dürrenbach, and Ohlungen (district of Oberstritten).

Within the same period, 212 leases have been recognized as valid in the Zabern fault-area; oil has been struck at various horizons from the Trias up to the Oolite inclusive. It generally comes up in drops of a blackish-brown colour with the water used in the boring; seldom does asphalt occur in conjunction with the oil. The average depth at which the latter was struck was 82 feet, or only about a third of the average depth in the Tertiary area. At the first glance, therefore, one would be inclined to think that the Zabern oil-field is more favourably situated than that of Pechelbronn, and yet no regular winnings of petroleum have so far been undertaken in the former district.

The greatest depth yet reached by any boring in the Tertiaries is that of Oberstritten—2,034 feet. No oil was struck, but proof was obtained of a hitherto unimagined thickness of Lower Oligocene marls, and interesting observations were made on the increase of temperature with the depth below the surface.* Prof. Daubrée here makes a mistake, due to an error, in declaring that the thermal interval or number of metres for the increase of each degree diminishes in the direct ratio of the increase of depth.

A circumstance worthy of notice is the frequent occurrence of the oil in association with mineral waters, which contain generally a high percentage of bromides. These waters, however, are often found to occur alone, and their saline content is very variable.

The last point to which particular attention may be directed concerns the relations of the respective borings one to the other. Of great importance, in this connexion, are the three borings, Nos. 146, 186, and 213, in the neighbourhood of Pechelbronn. At approximatively identical depths, namely, 443, 452 and 459 feet, they have yielded abundant springs of water and oil of the same character. There is no doubt whatever that these are derived from one and the same deposit, and a line drawn through the three borings exactly corresponds to the average strike of the sands formerly worked at Pechelbronn. There is a similar correspond-

* Prof. Daubrée, *Comptes Rendus* for 1893, page 265; *Zeitschr. für praktische Geologie*, 1894, page 69; and *Trans. Fed. Inst.*, vol. vi., page 594.

ence in the case of the line connecting spring No. 344 on the northern edge of the Hagenau forest with spring No. 346 at Surburg. One sees then that the deposits tapped by boring have the same strike as those which were formerly worked by mining, and in continuing exploration-work this parallelism should be borne in mind. In 1885, Bergrath Jasper first drew attention to this general coincidence of strike, but he made it run too much easterly.

In view of the above-mentioned dependence of the respective borings, it will be seen that rich springs may be suddenly dried up by the putting down of new search-holes. This occurred, for instance, in the case of boring No. 334, which drained No. 337 in the same, and No. 9 in a neighbouring, property.

The production of crude oil from Lower Alsace in 1894 was 15,632 tons: the refined petroleum produced amounts to 4,000 tons per annum, or 1·3 per cent. of the consumption of Germany. The yearly imports into the German Empire from America average 250,000 tons and from Russia 50,000 tons.

4.—The Origin of Petroleum.

We now turn to the consideration of this question, most of the views upon which will be found summarized in Mr. Höfer's book, *Das Erdöl und seine Verwandten*, Brunswick, 1888, pages 101-132.

Mr. Strippelmann held that in Alsace the original *locus* of the hydrocarbon compounds, whence petroleum is derived, should be looked for in the older strata—Silurian, Devonian, and Carboniferous. The transference of the deposits from older to newer strata took place in the earlier ages of the earth's history, and the oil-bearing deposits in the Tertiary marls must be regarded as complete closed reservoirs, which can receive no further supply of gas or fluid from the original stratum. Fissures of volcanic origin, whose general direction is parallel to the Vosges (the line Lobsann, Pechelbronn, and Schwabweiler forming a cross fault) facilitate the upward and outward movement of the oil. Messrs. Pochon and Venator are of entirely the same mind as Strippelmann.

Mr. Mieg remarks that a vegetable origin, that is, the decomposition of plants, is difficult to reconcile with the enormous pressure of the gases which accompany the oil.

Mr. Piedboeuf agrees in the main with Mr. Strippelmann, but he argues that the original stratum cannot be lower down than the Muschelkalk, because no trace of oil has been found in the porous Bunter Sandstone below, which should have furnished an excellent reservoir. He

thinks that the petroleum arises from the decomposition of animal matter, and more largely from fæces than from the animal organisms themselves.

Mr. Andreae, on the other hand, holds that the original petroleum-bearing stratum belongs to the Tertiary, pointing out that both in Upper and Lower Alsace the oil occurs at a particular horizon. He attaches no weight to isolated occurrences in other strata. He thinks that we are dealing with a lagoon and delta-formation, and that with the decomposing plants which have yielded the brown-coals, as well as the bitumen and oil, were mingled animal remains, to which the nitrogen content of the oil is due.

Mr. Achille Le Bel thinks only two views worth considering, namely, that the oil was derived from distillation of coal-deposits at great depths by the earth's heat, or, as Prof. Mendelejeff holds, through the action of water or of water-vapour on carbonaceous iron within the glowing interior of the earth. He inclines more to the latter view. Indeed the former view does not fit in with the geological facts.

Prof. Mendelejeff's theory appears to gain additional support from Mr. Engler's contention that the high carbonic oxide content of the Pechelbronn natural gases (3·15 per cent.) is contrary to the hypothesis of a kind of fermentative origin, and accords more with the conditions of dry distillation. Prof. Daubrée's opinion seems to take much the same direction as Prof. Mendelejeff's.

Bergrath Jasper, in conducting subterranean workings at Lobsann, made observations which tend to show that migration of the oil through the clefts in the strata, and permeation of other strata may still be taking place. But his views as to the original petroleum-bearing stratum are nowhere very definitely expressed.

Mr. Schumacher holds that petroleum and asphalt-deposits probably arise from chemical decomposition of animal remains heaped up in lagoon or delta-formations.

The above summary has shown how various are the views regarding the origin and formation of petroleum. The writer agrees with Messrs. Andreae and Schumacher that it was originated in Tertiary times, and one of his main reasons is the same as that set forth by the first-named author, namely, that in Lower as in Upper Alsace the oil is found associated with the same strata (chiefly the upper portion of the Lower Oligocene). No other view is supported by so weighty an argument.

Another main reason is the intimate connexion between the brown coals and the petroleum, and the abundant occurrence of *Helices* (snails)

in the boundary-zone. Still further reasons may be cited in support of the view that the original petroleum stratum belongs to the Tertiary.

For instance, it is by no mere chance that the occurrences of oil in Upper and Lower Alsace lie in the direction of the synclinals which separate the Vosges from the Haardt in the north and from the Jura in the south.

Of considerable importance also is Prof. Daubrée's observation that the oil-bearing strata contain a particularly high percentage of pyrites, very finely disseminated. Most of the sulphur in this is probably derived from albuminous compounds, and its abundance points to the former presence of large masses of organic matter.

Moreover, the high gaseous content, especially of the deeper oil-deposits, is more easily explicable if we think of the oil as having been formed *in situ*. It is true that gas alone is got from some wells, but the writer thinks this only proves that there has been a partial displacement of the oil-deposits within the Tertiary itself.

The association of oil and saline springs needs no explanation, if we connect them with the deposition of the Tertiary brackish-water strata.

At one time it was denied that petroleum could be got out of organic substances, except at an intense glowing heat. What little foundation there may have been for this objection was proved by the recent researches of Mr. Engler, who succeeded in producing petroleum from animal fats.

Much emphasis is usually laid on the varying character of the bitumen concurrently with the depth from the surface ; thus the deepest strata of Schwabweiler yield the lightest oils, Pechelbronn heavier oils, and the more recent beds of Lobsann viscous asphalt. This is generally explained by the hypothesis that the oil nearer the earth's surface becomes oxidized, and is deprived of its more volatile constituents. But it must be borne in mind that the very deep boring at Oberstritten yielded a viscous bitumen ; and there is some plausibility in Mr. Le Bel's theory that the differences are those of original composition, and not of subsequent alteration. This would constitute one more argument in favour of the original petroleum stratum belonging to the Tertiary.

It is true that oil has been got at many localities in the Mesozoic area, but this does not at all imply permeation from below. It must be remembered that many horizons of the Mesozoic in Lower Alsace are extremely rich in fossils, and thus contain within themselves the means of giving rise to bituminous products. Thus the *Gryphæae* of the Gryphite-limestone are highly bituminous, while in Kraichgau petroleum

has been found filling the chambers of ammonites. The fish-bearing *Posidonia*-shales of the Upper Lias contain so much bitumen that when heated in air they burn with a vivid flame, or if air be shut off, yield an abundant bituminous distillate. No great quantities of petroleum, however, appear likely to be got from the Mesozoic of Alsace. It is doubtful whether even the *Posidonia*-shales, if treated by dry distillation, would repay working: the experiments made under similar conditions in Suabia are not encouraging.

Another fact which tells against the theory of impregnation from below is the repeated interbedding of very thin bands of oil-bearing and barren strata at Schwabweiler.

In favour of the theory of emanation from great depths through fissures, Prof. Daubrée adduces the abnormal relations of temperature observed in the Oberstritten boring; but it so happens that the temperature does not increase regularly downwards, in fact the increase is very small indeed. The greatest variations from the known figures are shown by the middle depths—thus at 1,082 feet we find a temperature of 126·5 degs. Fahr., whereas at the normal rate we ought to find only 87·4 degs. Fahr. Similar results are obtained from the Oberkutzenhausen boring, and it is a point worth discussing whether the source of the heightened temperature is not to be found in the higher strata themselves.

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Mr. Boverton Redwood (London) said that last autumn he had visited the oil-fields of Lower Alsace, and some of the information which he had collected might perhaps form a useful supplement to the interesting paper to which they had just listened. He might, perhaps, in the first instance remind them that this oil territory was not the only one in Germany; there were in fact three districts in which petroleum occurred, namely, (1) Wietze, Steinforde, Oelheim, and elsewhere in Hanover; (2) Lobsan, Schwabweiler, Pechelbronn, and elsewhere in the neighbourhood of Hagenau (16 miles north of Strasburg), in Lower Alsace, including Altkirch (32 miles south-south-west of Colmar), not far from Mühlhausen, in Upper Alsace; and (3) the district on the western side of the Tegernsee in Bavaria. Of these the Alsace district was, however, by far the most important. Here, in addition to the property of the Pechelbronn Company, a large number of concessions had recently been granted, and the demarcated oil territory now extended in the aggregate over 19,000 acres (720 square kilometres). Over this immense area, up to the present time, comparatively little drilling had been carried out, but as the government granted no concession until reasonably satisfied of the occurrence of petroleum on the lands demarcated, it might be assumed that there was a very considerable area of valuable oil territory. Each concession was for an area of about 1 square mile, and was held direct from the German government in perpetuity on payment of £1 (20 marks) per annum, and 2 per cent. of the net profits from working. On some of the concessions the surface-indications of the occurrence of petroleum were held to be sufficient to justify the grant of the concession, while in other cases the applicants were called upon to make tests by drilling, and in the latter instances oil was met with at depths ranging from 150 to 300 feet. The drilling of the wells on the Pechelbronn property had usually been conducted on the water-flush system, which appeared to give very satisfactory results in the comparatively soft formation overlying the oil-deposits, and that district was one of the few in which this system

of drilling, as applied to petroleum, could be seen in successful operation. It was asserted that by this system as much as 25 feet per diem could be bored. In the water-flush system, hollow rods, and a bit with a hollow stem and orifices in the shoulder, were employed, and water being pumped through the bit and overflowing at the mouth of the well, the detritus was thus continuously removed from the borehole. It was a system not found to answer in all cases, but in the sand formation met with in Lower Alsace it was expeditious, economical, and, on the whole, had given very good results. In some instances, where somewhat harder rock than usual was met with, the free-fall system of drilling had also been employed. The oil was found in sand, interstratified with marl, rather than in compact sandstone. On the Pechelbronn property, the flowing-wells which had been obtained were stated to have given an average yield of from 10 to 15 tons in twenty-four hours, one-third of this being water, and two-thirds oil. He observed that in some of these flowing-wells much gas, having a slight odour of sulphuretted hydrogen, accompanied the oil, but the latter was asserted to be free from sulphur. Sulphur was a most undesirable impurity, but he was assured that although the natural gas was occasionally contaminated with sulphur compounds, the oil was practically free from them. Some of the wells were said to have yielded an undiminished quantity of oil for the past ten years. On the adjoining Rudolph concession, drilling was commenced in 1889, and recently the Canadian system of drilling (which differs from the free-fall system in many respects, but mainly in the substitution of "jars" for the "free-fall" appliance) had been successfully introduced by Mr. G. S. Crosbie, a driller of considerable experience in the Galician oil-fields. The Canadian system was less expeditious than the water-flush system, but it was better adapted for dealing with hard strata, and Mr. Crosbie informed him (Mr. Redwood) that it answered satisfactorily in Alsace. In some cases, Mr. Crosbie had, for the sake of saving time, commenced with the water-flush system and completed with the Canadian system, and one well that he showed him (drilled in that manner last year) yielded about 70 barrels a day. Much of the oil occurred in this district in sand interstratified with marl rather than in compact sandstone, but in some portions of the territory Mr. Crosbie had met with a compact rock overlying the oil-bearing formation, and where this was the case the surface-water could be effectually shut off. In most cases, however, it appeared to be impossible to exclude water from the borehole, and he was disposed to regard this as the principal difficulty which

had to be encountered. He observed that considerable quantities of water accompanied the oil obtained on the Pechelbronn property, and he therefore could not endorse Dr. van Werveke's remark that "one great advantage of the Lower Alsace oil-wells is that the oil in them is mixed with comparatively little water." If they had a formation which was not provided with hard strata, it was impossible to prevent water passing down, to a large extent, to the oil-bearing beds, and they might have in Lower Alsace a state of things occurring such as happened at Oelheim, in Hanover, and led to the abandonment of that territory. There the water was not satisfactorily excluded, and the result was that it passed down in increasing quantities to the oil-bearing strata, until for every barrel of oil taken out, something like 90 barrels of water had to be raised, and so it was found impossible to carry on the industry. The matter would perhaps be better understood if they took into consideration the procedure in the United States, where an oil-well was usually described as consisting of three sections—the first formed of surface-clays and gravels; the second of stratified rocks containing water, and the third of stratified rocks, including the oil-sands, usually free from water. By the use of casing, the water in the two upper sections was excluded from the well, and the oil was usually obtained practically free from water. This could not be accomplished unless there were a hard impervious rock, immediately overlying the oil-bearing formation, in which a shoulder could be formed, to receive the lower end of the casing, or in which a watertight joint could be otherwise made, and it was the absence of such cover-rock in Alsace which stood in the way of the exclusion of the water. According to Prof. Engler* the specific gravity of the petroleum of Pechelbronn ranges from 0·878 to 0·885, but a sample from a well at Ohlung, on the Rudolph concessions, which he (Mr. Redwood) recently examined was of still lower specific gravity, 0·873. This sample had a flashing-point of 37 degs. Fahr. (Abel test), yielded 26 per cent. of kerosene, of specific gravity 0·799, and flashing-point 73 degs. Fahr. (Abel test), and contained a considerable proportion of solid hydrocarbons (paraffin). The crude oil was black by reflected light, and exhibited very little fluorescence, but in a thin layer by transmitted light it was of dark brown colour. The petroleum of Pechelbronn was examined by Mr. Boussingault in 1837,† and in recent years by Prof. Engler and others. According to Dr. Veith,‡ the thermal efficiency of the petroleum of Lower Alsace was as follows:—

* *Dingler's Polytechnisches Journal*, vols. cclxvii. and cclxviii.

† *Annales de Chimie et de Physique* [2], xliv. ‡ *Das Erdöl.*

Origin.	Specific Gravity	Water Evaporated per Unit of Fuel.	Effect in Heat Units
Pechelbronn	0·912 ...	14·30 ...	9,708
,, ...	0·892 ...	14·48 ...	10,020
Schwabweiler	0·861 ...	15·36 ...	10,458

There was a refinery of considerable size at Pechelbronn, where the crude oil was converted into the usual commercial products, and, judging from the samples shown him (Mr. Redwood) by Mr. Paul de Chambrier, director of the mines, the processes of refining appeared to be successfully conducted. The various operations and the products obtained had been fully described by Mr. Pochon.* The distillation was conducted in vertical wrought-iron stills, with hemispherical cast-iron bottoms, and domes with swan-necks. The stills took a charge of 10 to 15 tons (10,000 to 15,000 kilogrammes) of crude oil. The distillation was conducted very slowly, from ten to fifteen days, according to the size of the still, being occupied in working off a single charge. Superheated steam was employed in addition to direct fire-heat, and the distillation was continued to dryness, the residue of coke representing from 2 to 7 per cent. of the charge. As regarded the question of the origin of petroleum, he (Mr. Redwood) wished to say that in his opinion it was unwise to attempt to make any one theory apply to every case. The interesting researches of Prof. Engler had furnished much valuable testimony in support of the theory of animal origin which he and Prof. Höfer had so ably advocated. In respect to some descriptions of petroleum, such a theory might readily be accepted. For example, the nitrogenous petroleum of California described by Prof. Peckham, a pool of which, on exposure to the air, became filled with maggots, was no doubt of such origin. On the other hand, it appeared probable that some other varieties were mainly of vegetable origin. Prof. Phillips† had recently expressed the opinion that the slow decay of vegetable matter, under water, in the absence of air, was the only known process which yielded a gas at all similar in composition to the natural gas of the petroleum fields. He took exception to Prof. Mendelejeff's theory, on the ground that no free hydrogen such as would be produced by such a process was found in natural gas. In regard to the Höfer-Engler theory, Prof. Phillips had pointed out that the gaseous products obtained by Prof. Engler in the distillation of Menhaden oil and oleic acid contained from 25 to 38 per cent. of carbon monoxide, and a considerable proportion of olefines, while the amount of paraffins present was comparatively small,

* *Bull. Soc. Industrie Minérale*, 1893.

† *American Chemical Journal*, vol. xvi.

and he had stated that it was difficult to understand how such a gas could have been so changed in composition as to be rendered similar to natural gas. To those who were desirous of studying the geological conditions under which petroleum occurred, and the methods adopted in producing and refining it, he (Mr. Redwood) would strongly recommend a visit to Pechelbronn. The records of the work done were admirably kept, afforded full information as to the nature of the formation, and were generally of a most instructive character.

Mr. A. L. STEAVENSON remarked that in discussing the origin of petroleum the writer of the paper said, "Mr. Achille Le Bel thinks only two views worth considering, namely, that the oil was derived from distillation of coal-deposits at great depths by the earth's heat, or, as Prof. Mendelejeff holds, through the action of water or of water-vapour on carbonaceous iron within the glowing interior of the earth. He inclines more to the latter view. Indeed the former view does not fit in with the geological facts." The latest theory he had seen advanced was that it originated in the action of acid upon iron. Another point arose on the following remark: "In favour of the theory of emanation from great depths through fissures, Prof. Daubrée adduces the abnormal relations of temperature observed in the Oberstritten boring; but it so happens that the temperature does not increase regularly downwards, in fact the increase is very small indeed." He did not understand quite whether the writer disputed the fact that the temperature of the earth generally did increase as they got down, or only disputed it in this particular place. This point he thought was almost beyond dispute, and to raise such a question recalled an old saying that they might as well speak disrespectfully of the Equator.

The PRESIDENT said he would like to ask a question with regard to the water difficulty. As he understood, it appeared that the oils could not be obtained because there was no impervious bed met with in the borings; but if that was so, it was puzzling to him why the oil remained at the bottom and water at the top.

Mr. BOVERTON REDWOOD drew the following sketch, in which (*a*)

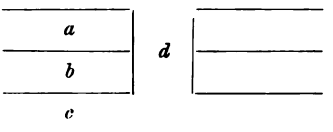


Fig. 1.

represented the superficial deposit charged with surface-water, while (*b*) was charged with salt-water; in drilling operations, they aimed at making

a junction between the iron casing in the well (*d*), and the capping-rock immediately above the oil-bearing formation (*c*). Having reached the impervious rock, the driller generally succeeded in making a shoulder to receive the lower end of the casing, and thus prevented the water from passing down to the oil-bearing formation. The President had asked why the water had not previously passed down to the oil-formation. He (Mr. Redwood) considered that it was because of capillarity and surface-tension; the water could not pass down until a certain amount of the oil had been withdrawn. When that occurred, there was a tendency for the water to enter and become mixed with the oil; but even then they did not find as a rule that the water displaced the lighter oil as might be supposed. But there was an increasing proportion of water with the oil flowing out of the well, and the difficulty he anticipated in working some of this territory was, as he had endeavoured to explain, the absence of that impervious capping-rock which played so important a part in enabling the driller to exclude water from the strata below. He thought there could be no doubt that, under such circumstances, there would always be a great deal of difficulty in obtaining the oil without a considerable admixture of water. With respect to the question of temperature, he took it that the point the author was referring to was that the increase of temperature actually observed in the wells at Pechelbronn was somewhat greater than would have been anticipated on the basis of the usual theory, and the only point he (Mr. Redwood) wished to mention was that this was by no means an unusual experience in a volcanic formation, or in districts which in past ages had been subject to volcanic action. He had one case in his mind, where the temperature was so high that the water used in the borehole began to emit steam, and although there was no means of ascertaining the temperature at the time, the increase was evidently abnormal.

Dr. Paul Dvorkovitz (London) said, after the remarks made by Mr. Redwood, he could only add some information received by him at the time of his visit in the year 1891 to the Alsace district. The sample of oil which he took then was quite free from water. But it seemed to him from the remarks of Mr. Redwood that this was not so now. The Pechelbronn oil contains a large proportion of asphalt, and it is very important that no water should be mixed with it in course of drilling, as such water cannot be separated by simple heating, which method is used with American and Russian oils. The distillation of Pechelbronn oil is generally conducted in the following manner:—The crude oil is first submitted to a slow distillation, which is finished in ten to fifteen days, depending on the size of the apparatus. The result of this operation is

two fractions: (1) crude burning-oil distillate, and (2) crude lubricating-oil distillate, rich in paraffine scale, and (3) in the apparatus a very hard coke is left. The crude burning-oil distillate, after being treated by sulphuric acid and washed with soda, is submitted to a second distillation, whereby it yields:—

	Specific Gravity.
Petroleum spirit	0·685 to 0·700
Naphtha	0·715 to 0·725
Burning-oil	0·800 to 0·825
Gas-oil and lubricating-oil.	

The last oil is submitted to a low temperature, and after the paraffin scale is removed, the oil is refined, and a very good lubricating-oil is produced. The total loss during the above-mentioned operations is not less than 20 to 25 per cent. With regard to the author's statement that the oil is formed where it is found, all facts known hitherto entirely disagree with this statement, as most oils are found in sand beds, which seems to serve as conveying-pipes. The view that oil is filtered and distilled from the place of its formation to different strata is clearly confirmed in Alsace. There are three distinct oil-districts. In the north at Lobsana and Kleeburg pure asphalt is found; in the central district at Pechelbronn asphalt is found mixed with light-oils; and in the south at Biblisheim, a pure oil without asphalt is found. He (Dr. Dvorkovitz) quite agreed with the remarks of Mr. Redwood about the impossibility of explaining the different characters of the oils, if only one theory of its origin is accepted. He preferred to adopt the theory of the formation of oil from peat. His own experience in the distillation of peat had shown him (Dr. Dvorkovitz) that 7 to 8 per cent. of oil of a petroleum character was produced at a very low temperature. The presence of nearly 90 per cent. of marsh gas in the natural petroleum gas, the presence of a large percentage of nitrogen and oxygen in most of the oils could be easily explained if we adopt the origin of petroleum from peat.

Mr. EDWIN R. BLUNDSTONE (Westminster) wrote that the conditions under which Dr. van Werveke had found petroleum to occur in Alsace evidently closely resembled those which he (the writer) had himself observed in Roumania. There was close parallelism equally as to geological age, and as to the particular character of the containing rocks. He had no doubt, too, that when enterprise, equal to that which had so successfully opened up the petroleum resources of Alsace, was directed to the Roumanian oil-fields, an equally satisfactory result would ensue. The association of petroleum with mineral springs, alluded to by Dr.

van Werveke, was also very marked in Roumania; and, although he (Mr. Blundstone) had not heard of bromides in noticeable quantity in the Roumanian waters, he had found iodides to a considerable amount. Common salt, too, which here, as so often elsewhere, was associated with petroliferous strata, was sometimes highly charged with ammoniacal salts. In connexion with the interesting summary of the conflicting views at present more or less entertained as to the origin of petroleum, he (Mr. Blundstone) would venture to offer the following remarks:—He had always felt great difficulty, in endeavouring to follow the views of those who propounded an animal origin for petroleum, in seeing where are to be found those large stores of animal material from which petroleum was, or is to be, formed. Certainly the total tonnage of living animal matter was small comparatively speaking, and there was no reason to suppose that it was greater at any time in the past. Moreover, it should be borne in mind that death nearly always came to an animal as the result of combat, the corpse of the conquered being eaten by the conqueror. In other cases, the corpse was consumed by microscopic forms of life, and in any case it was certainly very rare for animal matter to be left to undergo a slow "chemical metamorphosis," this latter being, as the writer supposes, the imagined equivalent of those processes of destructive distillation, which have been pointed to as yielding oils more or less like petroleum. The same difficulties, as to lack of sufficient supply-material, do not beset those who favour a vegetable origin for petroleum. The normal decidua of living plants and the remains of dead ones, together aggregate an enormous tonnage, the mere bye-products of the decomposition of which could certainly furnish all known petroleum. Further, the decomposition of large accumulations of these decidua can be demonstrated in any bog, marsh, pond, or *Zostera*-bed; but where can the parallel case for animal remains be shown? Chemically, it was at least as probable that the petroleum hydrocarbons are the result of the breaking down, in the absence of oxygen, of the cellulose or cellulose-like molecule, as so often occurs in fermentations, as that they were formed from either proteids or their products, or from fats. With regard to Prof. Mendelejeff's theory, illustrious by virtue of its author's name, it was unsupported by any positive evidence, and it must strike everyone as curious that petroleum, formed, as the professor maintained, deep down in the earth, should never find its way through fissures in the very ancient rocks, where these latter outcrop, but always chooses for exit those fissures which were, unfortunately, overlain by much later rocks. Mr. F. C.

Phillips has recently published* an elaborate research dealing with the origin of petroleum. He pointed out that natural gas contained neither hydrogen nor carbon monoxide (the 3·15 per cent. of this gas found in Pechelbronn natural gas was a fact quite *sui generis*), and enlarged upon the difficulty of accounting for the absence of the former gas, if Prof. Mendelejeff's theory was to be held. Mr. Phillips had also produced practically the only gas which at all resembled natural gas, and had done so by allowing vegetable matter to undergo an extremely slow process of decay.

The PRESIDENT proposed a vote of thanks to Dr. van Werveke for his interesting paper, and he was sure their best thanks were also due to Mr. Boverton Redwood, Dr. Dvorkovitz, and others for the interesting information which they had afforded during the discussion.

Mr. H. W. HUGHES seconded the motion, which was cordially adopted.

The following paper by Mr. John McConnell on "The MacArthur Forrest Process" was taken as read:—

* *American Chemical Journal.*

THE MACARTHUR-FORREST PROCESS.

By J. McCONNELL.

The MacArthur-Forrest process, patented in 1887, is described in the specification as having for its object the obtaining of gold from its ores or other similar compounds, such as mattes and slags, and also as being applicable for the obtaining of silver from its ores or other compounds by the treatment of such ores or compounds with a solution of cyanogen or cyanide, or other substances containing or yielding cyanogen.

The cyanide generally used is cyanide of potassium, the process being based upon the fact of gold being soluble in a solution of that salt, forming with it a double cyanide of gold and potassium. The ore to be treated having been reduced to a fine state of division, is mixed without further treatment with a dilute solution of cyanide of potassium, and agitated or allowed to saturate with it until the gold is dissolved. The solution is then separated from the ore, and the gold precipitated by passing it through zinc turnings. The gold precipitate is carefully collected from the turnings, mixed with a suitable flux, and smelted.

The advantages of the process, both as regards efficiency and economy in the treatment of gold-bearing ores and tailings in general, are great, but more especially in dealing with refractory ores, which, owing either to their complex nature or to the gold existing in them being coated or rusty, or in an extremely fine state of division, cannot profitably be dealt with by other means. In complex ores containing compounds of iron, copper, zinc, lead, arsenic and antimony, cyanide of potassium, when a dilute solution is used, has little action on these compounds, as they are generally found in ores. Any action that may take place can be reduced to almost *nil*, by regulating the strength of the solution used in the treatment, except perhaps in the case of copper compounds, which, when met with existing in a soft or porous state, interfere very materially with the extraction of gold, and increase greatly the consumption of cyanide. Hard copper pyrites, however, is not readily acted on by a dilute solution of cyanide of potassium.

The writer has frequently treated some very refractory ores and concentrates containing compounds of iron, copper, zinc, lead, etc., and, by using a solution containing 0·35 per cent. of cyanide of potassium, he has obtained an average extraction of gold of over 90 per cent.

With regard to ores free, or practically free, of base metals which cannot profitably be dealt with by amalgamation, owing to the gold existing in them as coated gold, or in an extremely fine state of division, the cyanide treatment is most effectual and economical. The extraction of gold obtained from such ores is not only high, generally averaging from 92 to 95 per cent. of the assay value, but the consumption of cyanide in the treatment is very low, rarely exceeding 2 to 3 lbs. of cyanide of potassium per ton of ore treated. Frequently, especially in the treatment of tailings, even less cyanide than this is consumed.

At the start of the cyanide treatment, it was generally believed that only low-grade ores or tailings could be treated profitably by it. Since then, however, it has been proved otherwise. Very rich refractory ores can be treated profitably by the cyanide process.

The writer has treated with cyanide, ores from the Silverton mine, New Zealand, ranking as high as £400 per ton assay-value, and has recovered, at a small cost, bullion to the value of from 99 to 99·7 per cent. of their assay value.

When tailings containing pyrites to any great extent which have been exposed to the weather for a time, are to be treated, a difficulty is met with. Owing to the pyrites becoming more or less decomposed by action of the air, free sulphuric acid and soluble iron salts are formed. If such tailings are treated direct with cyanide, these compounds decompose the cyanide of potassium solution, and increase largely the consumption of that re-agent. It is necessary, therefore, to treat such tailings first with a water-washing, or, if required, an alkaline washing before treatment with cyanide, to leach out or neutralize such compounds. In some cases the lime or other alkali required is added with the cyanide solution.

When ores are met with containing similar compounds which act upon the cyanide solution, it is necessary to wash such ores also with water or alkali solution before the treatment with cyanide.

Methods of Treatment.—The method usually adopted for the treatment of ore by cyanide is percolation or, in other words, by allowing a cyanide solution to filter slowly through the ore. For this method of treatment, the plant required consists mainly of wooden tanks. The tanks for treatment can be of any size, depending on the quantity of ore to be

treated, and either round or square, preferably round and not too large. A few inches from the bottom of each tank is fixed a wooden grating. The gratings are supported by boards 1 inch thick and 3 inches broad, laid edgeways across the bottom of the tanks, and are covered with canvas-filters. The canvas-filters are wedged tightly between the edge of the gratings and the sides of the tanks so as to prevent any ore from passing through. If the ore after treatment is to be shovelled from the tanks, narrow strips of wood 6 to 9 inches apart are fixed along the top of the canvas to prevent it from being cut by the shovel. If possible, however, the ore after treatment is sluiced out of the tanks, this being a much quicker and less expensive method. If the tanks are to be emptied by sluicing, a sluice-gate is constructed on the side of each tank on a level with the top of the filter. On the outside of the tanks below the sluice-gates, a launder is constructed to carry the ore, when sluicing, clear of the plant to the tailings-pit.

In charging, the ore is tipped from trucks which run over the top of the tanks, and is carefully and evenly spread. The depth to which the ore is charged into the tanks varies from 3 to 6 feet, depending entirely on the nature of the ore, and how it will filter. If the ore is of a clayey nature, 3 feet is sufficient, but if it be free of clay or slimes, it can be safely charged to a depth of 6 feet, or even more. When sufficient ore has been charged into a tank, the cyanide-solution is run in by means of a pipe which enters the tank below the filter.

The solution is allowed to rise up from the bottom slowly through the ore until it appears a few inches above the top. It is then stopped. This method of running on the solution is adopted so as to saturate the ore evenly, and prevent channels being formed through it, as would be the case if the solution were run on to the top of the ore.

After the ore has been saturated with the solution for an hour or so, a draw-off stopcock is opened in the bottom of the tank, and the solution allowed to percolate through the ore. The solution dissolving the gold in its course through the ore is carried from the bottom of the tank by means of an iron pipe, and passed through the precipitating-boxes. The ore is then washed for the purpose of freeing from it the gold solution. Two washings are generally sufficient for this purpose. These washings are run on to the top of the ore, and run off in the same way as the first solution.

If the solution and washings do not percolate freely, they are drawn off by means of a vacuum. For this purpose an iron boiler is provided, and is connected by an iron pipe to the draw-off stopcocks of the treat-

ment-tanks. The air is pumped from the boiler by an exhaust pump, and the vacuum caused draws the solution off from the ore into the boiler.

The solution, after being drawn off, is run from a stopcock fixed in the bottom of the boiler into a tank underneath, and thence passed through the precipitating-boxes.

Treatment of ordinary ore by percolation takes about two days, so that with four treatment-tanks, each capable of holding a charge of 20 tons of ore, 20 tons of ore per day can be dealt with.

If the whole of the output from a mine is to be treated by the cyanide process, it is preferable to crush the ore dry. By so doing, float-loss is prevented, and the ore is much more easily handled. There is no objection, however, so far as the cyanide treatment is concerned, when a wet-crushing plant is already erected, to treat the ore first by amalgamation and the tailings afterwards with cyanide.

Another method of treatment by the MacArthur-Forrest process consists in agitating the ore with a cyanide solution until the gold is dissolved.

For treatment by this method, convenient-sized wooden cylinders or tubs fitted with revolving paddles are generally used.

A cyanide solution is made up to the required strength, and first run into the cylinder, and while the paddles are revolving the charge of ore is gradually added. The ore and solution are agitated together until the gold is dissolved, which generally takes from four to six hours, although in cases when particularly refractory ore is being treated an agitation of twelve hours is sometimes necessary. The length of agitation required can only be determined by experiment. When the gold is dissolved, the charge is run into a filtering-tank, and the solution drawn off by means of a vacuum as already described. As there are few classes of ore which cannot be treated by percolation, the agitation process is not usually adopted. Power and machinery being both required for it, its working is more complicated and costly.

Solutions.—With regard to the strength of cyanide solutions used, there is no fixed rule. The strengths of solutions are varied according to the requirements of the ore to be treated. The proper strength for the treatment of any particular class of ore is only arrived at by experiment. For ordinary ore, a solution containing about 0·25 per cent. of cyanide of potassium is used, and the ore is generally treated with about one-half its weight of solution.

proportion of soda must be added. Too much soda, however, is objectionable, as it destroys the crucibles. The precipitate is mixed thoroughly with the flux and charged into the crucible. As the mixture fuses and runs down in the crucible, the slag is skimmed off, and more mixture added from time to time, until there is sufficient melted bullion in the crucible to be poured into a mould. The pieces of bullion thus obtained are melted together again with borax and moulded into bars of convenient size. Bullion recovered by cyanide, if care be taken, need not contain more than from 4 to 6 per cent. of base metal.

The slag obtained in smelting the bullion is preserved. It generally contains a small quantity of bullion. As this bullion is in the form of small beads, it can be most conveniently recovered by crushing the slag in a mortar, and panning it off. Where large quantities of slag are to be dealt with, it is advisable to obtain a small prospecting-stamper for crushing. By allowing a flow of water to pass through the stamper-box while the slag is being crushed, the beads of bullion can be caught on ripples, and afterwards collected and melted together.

Prof. A. W. Rücker read the following paper on "The Magnetic Survey of Great Britain":—

THE MAGNETIC SURVEY OF GREAT BRITAIN.

By A. W. RÜCKER, M.A., F.R.S.

The object of a magnetic survey is twofold: firstly, to determine the secular changes in the declination, horizontal force and dip; and, secondly, to investigate the local magnetic disturbances by which these elements may be affected.

In the years 1884-1888, the author and Dr. Thorpe, F.R.S., made magnetic observations at 205 places in the United Kingdom. The results were considered sufficiently interesting to make it desirable that the work should be extended. An allowance from the Government grant in aid of research enabled them to carry out this extension with the help of assistants, Messrs. Briscoe, Gray, and Watson. With the aid of these gentlemen, the magnetic constants have now been determined at 882 places in Great Britain and Ireland. The two surveys have been worked up independently, so as to give the magnetic condition of the United Kingdom on January 1st, 1886, and January 1st, 1891. The secular change between these epochs has been calculated, and all the observations have been finally reduced to the later date. By means of the table of secular change it will be possible to determine the normal, undisturbed value of the declination at any place in the United Kingdom for some years to come, and by means of a map on which the deviation from the normal values is shown at 882 places, it will be possible to gain some idea how far local disturbance is likely to affect the declination at any given place (Fig. 1, Plate XXI.). Similar tables and maps are provided for the other magnetic elements.

As regards local disturbances it is found that the north pole of the compass is attracted to certain regions and to lines which can be traced for scores or even for hundreds of miles. These lines are called magnetic ridge lines. As a general rule, a region where basaltic rocks appear in large quantities on the surface attracts the north pole of the magnet. On the basalt itself the disturbances are irregular, but an attraction towards it is observed at stations near to, but not actually on the magnetic rock. This is very clearly shown in Antrim (Fig. 4, Plate XXIII.), in Scotland, between the Forth and the Clyde (Fig. 5, Plate XXIII.), and in North Wales (Fig. 2, Plate XXII.)

manent magnetization is, however, irregular, and at a comparatively short distance the disturbing effect appears to be due almost exclusively to the uniform magnetization produced by the earth's magnetic field.

The author has made calculations on this point, based on the measurements of the magnetic properties of basalts above referred to, and during the magnetic survey special observations were made near isolated and prominent masses of trap. The general conclusions, both of theory and experiment, are :—(*a*) That dykes and thin uniform basaltic sheets produce no measurable effects except at distances from their edges, which are small multiples of their thickness ; (*b*) that isolated masses of trap-rock a few square miles in area produce no important magnetic effects at distances comparable with their linear dimensions.

Of course isolated cases may, and probably do, occur where one or other of these indications would be incorrect. They hold good in every case which has been specially investigated in the United Kingdom, such as the Island of Canna near Skye, Boulmer dyke in Northumberland, Titterstone Clee Hill in Shropshire (Fig. 6, Plate XXIII.), and Brent Tor in Devonshire.

The last point on which a few words may be useful is :—

III.—The Effects of Geological Faults on the Compass.

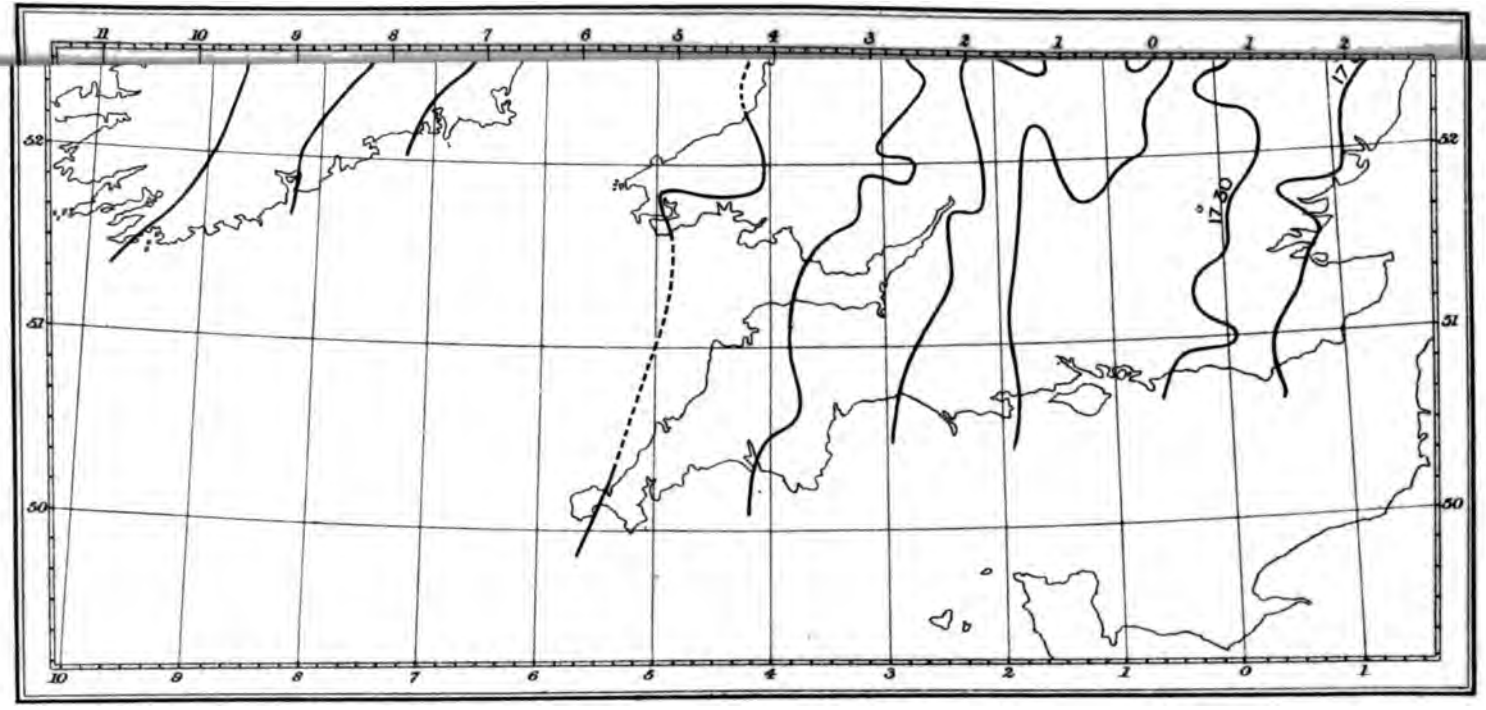
The view commonly held that faults affect the needle may in some cases have some foundation in fact. A line of magnetic attraction coincides very closely with the great fault marked by the Caledonian canal (Fig. 2, Plate XXII.). But though it is possible that magnetic rocks may lie nearer to the surface on one side of a fault than on the other, and that thus a magnetic disturbance may be produced, such an effect will be very small except under favourable conditions. If the throw of the fault is small compared with the depth at which the magnetic rocks occur, the compass-needle will not be affected.

It is hoped that, apart from the purely scientific interest of the determination of the positions of the lines and regions which attract the north pole of the magnet, such facts and generalizations as those above indicated may be of some practical use.

Prof. A. W. Rücker (London) believed that Great Britain was now the most fully magnetically-surveyed kingdom in the world. The lines of magnetic disturbances are very irregular in form, instead of being, as generally shown on maps of the world, smooth curves (Fig. 1, Plate XXI.). The most interesting fact was that a number of lines

Transactions [illegible]

To illustrate Mr A.W. Rücker's Paper on "The Magnetic Survey of Great Britain."



To illustrate Mr A.W. Rücker's Paper on "The Magnetic Survey of Great Britain."

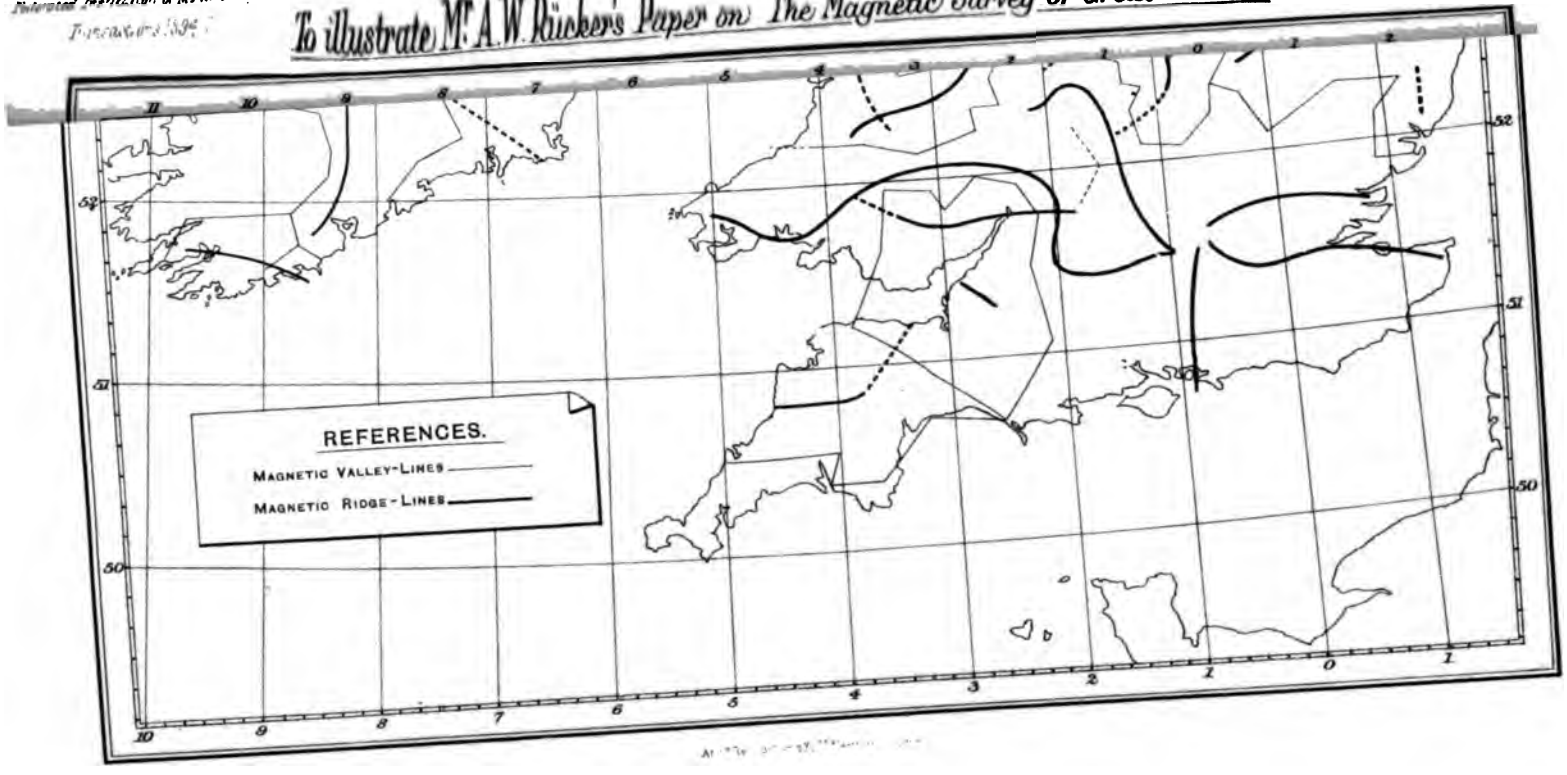
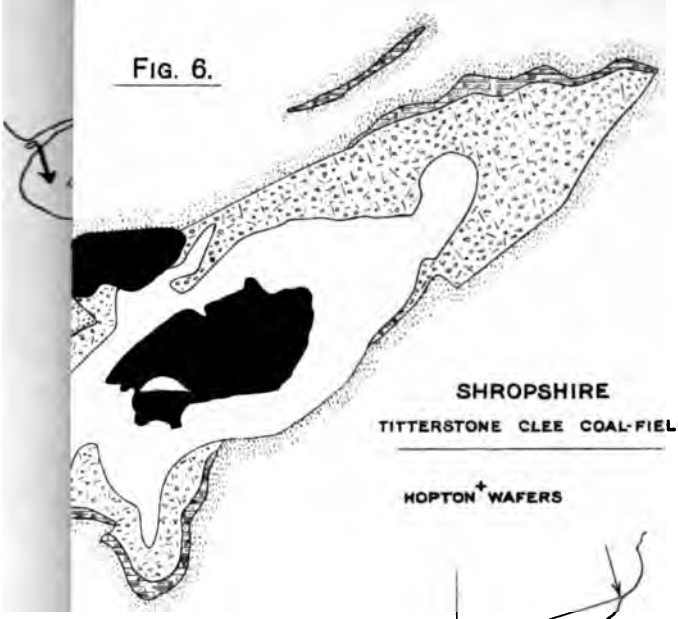


Fig. 6.



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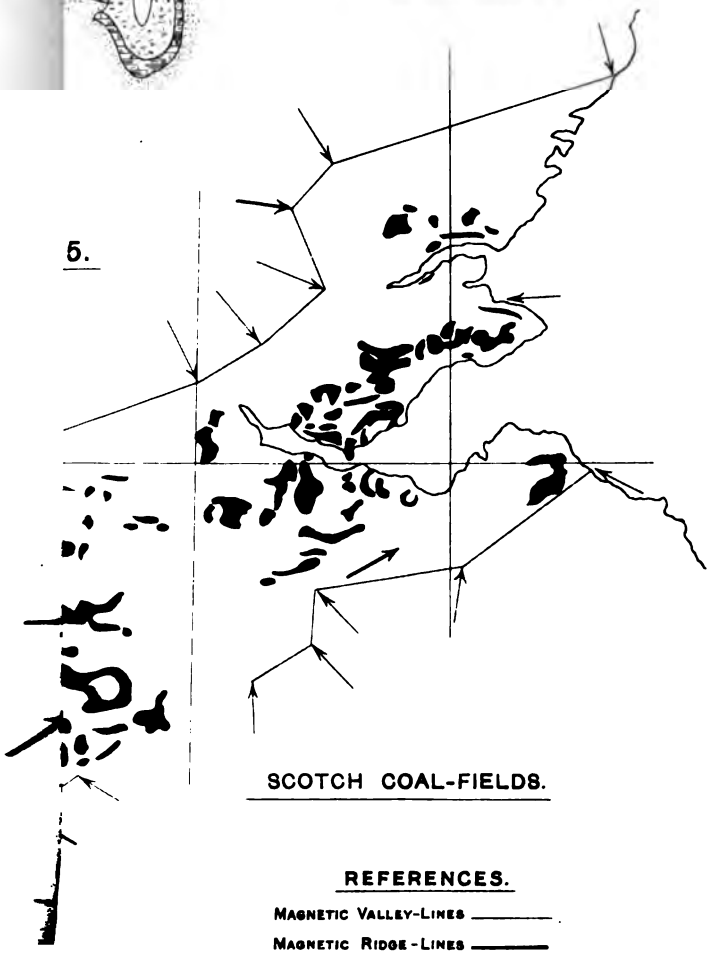


SCOTCH COAL-FIELDS.

REFERENCES.

Magnetic Valley-Lines _____

Magnetic Ridge-Lines _____

(ridge-lines) ran across the country, towards which the north pole of the magnet was attracted, and other lines (valley-lines) from which it was repelled. The second map (Fig. 2, Plate XXII.) showed the ridge-lines as thick lines and the valley-lines as thin lines, and at a glance one could ascertain the magnetic variation in any part of the kingdom. One of the interesting points connected with this survey was the fact that the ridge-lines ran across parts of the country where there was no magnetic matter to account for it, therefore they could only suppose that they were caused by concealed magnetic matter. The diagram (Fig. 3, Plate XXIII.) showed the ridge-line which ran across part of the country where there was no magnetic matter on the surface—across Lancashire and Yorkshire. The hypothesis they put forward was that, at a depth greater than that with which geologists dealt, there was concealed magnetic matter. The fourth diagram (Fig. 4, Plate XXIII.) represented the basalt which came to the surface in Antrim, Ireland, the disturbing forces being shown by the arrows. Coming to the points in which the members of the Institution would be more specially interested, he had drawn up the paper so as to give a description of the ridge-lines running across the coal-fields of the country. The Scottish coal-field was one of the most interesting, as there was much basalt and the magnet was attracted (Fig. 5, Plate XXIII.). The only other point was the effect of masses of basalt in producing magnetic disturbances. If the observations were taken close to a magnetic rock, the compass might be affected by many degrees, but the effect diminished very rapidly with the distance. The following table shows the effects of a mass of basalt in different latitudes, and it might be said in this country that at a distance from a mass of basalt, comparable with its own breadth, no important influence would be experienced.

TABLE SHOWING THE DISTURBANCE DUE TO A RECTANGULAR MASS OF BASALT. LENGTH, 1; BREADTH, 1; AND DEPTH, $\frac{1}{2}$, AT A DISTANCE FROM EDGE OF 1.

Place.	Latitude.			Disturbance.	
	Degs.	Mins.		Mins.	Secs.
Pawlosk	59	41	North	4	20
Los Angeles	34	3	North	3	30
Batavia	6	11	South	2	48
Spitzbergen	78	31	North	49	0

Prof. G. A. LEBOUR (Newcastle-upon-Tyne) asked whether Prof. Rücker had made any observations respecting the magnetic influence of

the whin sill of Northumberland and Durham. The whin sill, being so very extensive a sheet of magnetic rock, very probably exerted an influence far outbalancing that of mere dykes, and at Bamburgh and Middleton-in-Teesdale—where it formed large surfaces—that influence must, he should imagine, be of considerable importance, and quite sufficient to invalidate surveys carried out without due care in using of elaborate corrections.

Capt. E. CREAK, R.N. (London) said that the two magnetic surveys of the British Isles conducted by Profs. Rücker and Thorpe for the epochs 1886 and 1891, were the most complete that he was acquainted with amongst the many magnetic surveys of other countries which had been made. No details which could minister to the exact knowledge of the magnetic lines of magnetic declination, dip or force, had been omitted. One point stands out among the many in the interests of The Federated Institution of Mining Engineers, viz., that Messrs. Rücker and Thorpe have shown the connexion that exists between the disturbances from the normal of the magnetic needle and geological formation, and he would call special attention to this point. He was not aware to what degree of accuracy the magnetic declination was required for mining operations, but he should assume from the results obtained by the author of this paper that the compass in many parts of this country was an unreliable instrument where great accuracy was required. Fortunately for the British Isles, the areas of magnetic disturbance are not as great as in some countries; as in New Zealand, for instance, where during the naval survey the compass had to be discarded and true bearings alone could be utilized on land. It should be also remembered that the magnetic surveys hitherto made have been confined to the surface of a country, and we have but meagre information of what may be experienced at the depth of many mines. He was, therefore, inclined to think that in all cases where possible true bearings by the theodolite should be employed. Prof. Rücker showed that with rare exceptions the north end of the compass-needle was attracted towards the ridge-lines of his survey. This was true of ridge-lines in the northern hemisphere, but all observation at present went to show that in the southern hemisphere the north end of the compass-needle was repelled. Thus, the Transvaal miner would almost surely find the north end of his compass repelled from any ridge-lines that might exist. He had only once more to express his great appreciation for Messrs. Rücker and Thorpe's work; but the very excellence of that work had made him more than ever doubtful of the use of the compass for surveying purposes.

Prof. H. STROUD (Newcastle-upon-Tyne) had pleasure in bearing testimony to the very great value of the complete magnetic survey of the

United Kingdom recently carried out by Messrs. Rücker and Thorpe. In the present paper, the detailed magnetic maps, giving the lines of disturbance and showing their relations to the coal-fields of the country, would prove of great interest to all to whom an exact knowledge of the magnetic declination at a given place at a given time was of practical value.

Prof. A. W. Rücker said that Prof. Lebour thought that the whin sill must produce a discrepancy. Observations which he (Prof. Rücker) had made showed that even in the whin sill district, although they were not far from the rocks, they would find no effect, and they must remember that if a sheet of rock were magnetized in the earth's field both surfaces were magnetized oppositely. It was only when they went close to the edge that they would find any effect at all. Captain Creak referred to the Transvaal, and said that the north end of the needle would be repelled from any ridge-line. This was true in the southern hemisphere; the rocks of the ridge-lines attract in the northern and repel in the southern hemisphere.

The President said that they were much obliged to Prof. Rücker for his kindness in addressing them on the subject. Fig. 6 (Plate XXIII.) was a map of a small coal-field in Shropshire, where the basalt lay on the top. He should be rather interested to know if there were any particular effects observed in that locality.

Mr. W. F. Howard (Chesterfield) said that the members knew that the magnetic needle was not a reliable indicator, but he had found in practice that very successful and satisfactory results could be obtained in survey by its use, by taking great care and precaution, by repeated observations on the surface and underground, and by checking in every possible way what was not absolutely permanent. It was, of course, a very great convenience to use the compass in mining surveys, and he thought with precautions it might still be of considerable value, though he had no doubt the time was coming when, with the greater extension of coal-fields and of workings underground, it would be necessary to adopt something more perfect than the magnetic needle.

Mr. Wm. Thomas (Camborne) said that in Cornwall they found that the declination varied very much, and even in such cases as cross-cuts driving through granite in a comparatively straight line they found that occasionally the needle varied as much as 2 degs. or 3 degs. at points only a few feet apart. There were some extraordinary cases in Cornwall, such, for example, as the Botallack mine, near the Land's End, where local attraction had been known to throw the needle 65 degs.; or South Tolcarne mine, where, in the course of making a survey, he had found a variation of 12 degs.

Prof. RÜCKER asked if that was an ironstone mine?

Mr. THOMAS said both were tin mines. The instances he had given were from underground experiences. At the surface, too, in the flat lode district, near Camborne, in the last few months it had been found absolutely impossible to get sufficiently accurate results with the needle. The needle seemed to be most unaccountably disturbed—perhaps it it might be by some of the contents of heaps of accumulated mine-waste on the surface, or the disturbing influence might have been underground. Anyway, that was the fact, and he thought that in the keeping up of many mine plans this important matter of local disturbance had not received the attention it required. In fact, too much reliance had, in many mining districts, been placed upon the needle, and very little trouble had been taken to ascertain whether local conditions rendered the needle more than ordinarily unsafe. Perhaps it might not be out of the way if he referred under that head to a practical section of the subject, viz., to the flooding of mines, due to holing into abandoned workings, of which no reliable plans existed. In many of these instances the disaster might be directly traced to the surveyor negecting to guard against errors caused by the declination of the needle. Such disasters caused heavy commercial losses, for the flooded workings were often permanently abandoned. Then large numbers of hands were thrown out of employment at a moment's notice. And, last but not least, there was often a heavy loss of life. There had been, during the last two or three years, three cases particularly which he was then thinking of, two in Cornwall and one in Staffordshire, and in those three cases of unexpected floodings 99 lives had been lost. Those facts were closely allied to the matter to which Prof. Rücker had drawn attention.

Mr. M. WALTON BROWN mentioned that the question in connexion with the Cornish mines had been dealt with in a paper read by Mr. Henderson on "Magnetic Declination in Mines," which was already published in their *Transactions*.* Mr. Henderson there stated: "It is a well-known fact that in parts of the Botallack mine the needle has been deflected to the extent of 50 or even 60 degs. from its proper direction, and the utmost care has to be used by the surveyor in consequence." Mine-surveying there must be done under difficulties.

The PRESIDENT said he should imagine the risk of mistakes being made was not so great when the needle was deflected to that extent as when it was deflected to the amount of 2 or 3 degs., as in the case mentioned by Mr. Thomas. There was no doubt that such an error would

* Vol. viii., page 273.

be quite sufficient in an extensive survey to make a material error, and when they considered the risk both to life and property which depended on the accuracy of mining plans, it was evident that the subject had a most practical side. Another point which struck him was as to what effect the whin dykes in the North of England, of from 10 to 60 feet in thickness and of unknown depth, might have upon the needle. Several of these dykes passed through the northern coal-field and also in other districts.

Mr. W. F. HOWARD said he would like to make one further observation. He thought it would be a mistake to suppose that in any mining districts the needle was relied upon to any very great extent. The fact of the matter was that they got one course thoroughly settled to start with, and from that bearing everything else was taken by angles, not by the magnetic bearings *per se*. It might be noted as a magnetic bearing, but in practice it was an angle taken—the survey was a traverse really. In the old days he had no doubt the magnetic bearing was taken by itself, and that there would then be much more liability to error than in the modern practice.

Prof. RÜCKER, replying to the remarks and questions, said that the diagram (Fig. 6, Plate XXIII.) referred to by the President, represented the geology of Titterstone Clee Hill, and it was shown because there they had an isolated mass of basalt, and it was interesting to determine at what distance that basalt would produce any effect upon the needle. The two places chosen for the observation were Bitterley and Hopton Wafers, about 1½ miles distant on each side, and they found practically no effect. That might be taken as a superior limit of the distance at which such a mass would produce any effect. He (Prof. Rücker) was the last man to put aside the practical application of scientific work as of no importance, but it would not be correct to say that that work was undertaken with a view to its application to mining. Their object was to determine the secular change, but in the course of their observations they found these lines of attraction running across the country, and they became important, not so much from their bearing on mining, as because they helped to form a picture of the geology of the underground parts of the earth at depths to which no geologist could penetrate. Without saying that the result was always due to that cause, he thought there were indications that those ridge-lines corresponded to great folds in the magnetic rocks, and therefore they hoped to get some idea of how those old strata were arranged. Coming to the practical question, if any practical use could be made of the

observations, no one would be more pleased than himself and his colleagues. With regard to the big disturbances to which Mr. Thomas referred, he well knew that disturbances of that magnitude existed. One very interesting case which had just lately been discussed was that of a disturbance affecting the magnet to an extent of 56 degs., which occurred off the north coast of Australia in over 180 feet of water. A disturbance of that magnitude at that distance was very exceptional, but if they went close to a basaltic rock, as he had been in Canna near Skye, they might get their needle to point in any direction they pleased when they chose their rock properly, but less considerable effects would be produced at some distance from them. In Black's *Guide Book* there was a picture of the island with a ship going to pieces off the rocks, which was supposed to be owing to the effect of the rocks on the compass. There was a place which, from that circumstance, was called Compass Hill. They put their yacht so as to sail within 600 feet of Compass Hill; then began to read the compass, and read it steadily every two minutes until within 600 feet of the hill. Of course, with the compass on board ship, they could not make their investigations with the same accuracy as on shore, but though on land they got deflexions of 40 degs., at 600 feet distant they could not detect any deflexion. All that they aimed at doing with regard to mining was to show the more important disturbances and the order of magnitude; they could tell what the character of the disturbance would be in a district, but they could not tell what it would be in the neighbourhood of a particular rock.

The PRESIDENT said he was sure they would agree in passing a cordial vote of thanks to Prof. Rücker for his valuable remarks. The point had been referred to with respect to the possible use to geologists of the conclusions arrived at by Prof. Rücker, and it almost appeared that it was a sort of resuscitation of the divining-rod on scientific lines.

Mr. W. THOMAS had great pleasure in seconding the proposition. He would like to see the use of the needle for mine-surveying entirely abolished, and Prof. Rücker's paper would prove exceedingly beneficial if it helped towards the attainment of so desirable an end.

The vote of thanks was passed by acclamation.

The following paper by Mr. Robert Oates on "The Copper and Tin Deposits of Chota-Nagpore, Bengal, India," was taken as read :—

THE COPPER AND TIN DEPOSITS OF CHOTÀ-NAGPORE, BENGAL, INDIA.

By ROBERT OATES.

The writer is not aware of any paper having been published on this subject in the *Transactions*, and he hopes that the following description, which comprises the latest particulars known, may be of interest.

Situation.

The immense copper range of Chotà-Nagpore is by far the largest known deposit in India: commencing at Lopso in the highlands of Chotà-Nagpore, it at first takes a north-easterly course, then gradually trends in a south-easterly direction to Bairagoora in the east, having a total known length of some 80 miles (Fig. 1, Plate XXIV.). It passes through the following rivers in its course:—Sanjai, Kherkai, and Gurha, besides innumerable nullahs (water-courses), all of which flow in a north-easterly direction into the Subanrekha river, which again runs in a south-easterly direction towards the Bay of Bengal.

Access.

This district has since the early part of 1891 been opened out by the Bengal-Nagpore railway, which starts from Asansol on the East Indian railway, and taking a south-westerly direction meets the Great Indian Peninsular railway at Nagpore, thus forming a second route from Calcutta to Bombay. The nearest station to the copper-range is Sini, some 225 miles from Calcutta. Two miles or so further on, or about 227 miles, the line passes through the copper-belt, 64 miles laying to the east of the line, and 16 miles to the west. The nearest old workings exist at Dugni, east of the railway.

Geology.

The copper deposits occur disseminated throughout a belt of schistose formation belonging to the Sub-Metamorphic period, and consist of bedded schists and quartzites, which again rest on a wide extent of gneiss, principally found in the centre of the copper-range.

The bedded quartzites, which lie below the schists, also have lenticular masses of copper-permeated schists interspersed throughout them (Figs. 2, 3, 4, 5, and 6, Plate XXIV.).

The whole of this belt has been upheaved from the south at an average angle of 40 degrees; the quartzites have formed broken hills, sometimes 1,500 feet in height, where the force of upheaval has been strongest. The principal hills proceeding eastward are: Dhoba, Rangamatia, Kappergadi, and Sideshur. All these hills are highly precipitous on the southern sides, and huge slabs of quartzite lie at the bottom, testifying to the rapid disintegrating action resulting from severe climatic conditions. The hills consequently, are more conveniently approached from their north or sloping side.

The small hills which form the spurs of these larger hills, are composed of schists and invariably form the copper-belt, the schists sometimes being replaced by quartzites, and *vice versa*, so that the outcrops take rather an erratic course. It will naturally follow that where these elevations occur, outcrops have been formed, and owing to their exposed condition, the copper pyrites has yielded all its sulphur, leaving strings, incrustations, and segregated masses of blue and green malachite. Where the atmospheric influences have not been so favourable, only a partial decomposition has taken place with black oxides as the result, and at a lower depth the original copper pyrites would appear, where atmospheric agency had had no effect.

Trap-intrusions are of frequent occurrence, and appear to have originated after the upheaval of the Sub-Metamorphic rocks. Wherever they show themselves, the schists have become almost unrecognizable. The dip and strike have not suffered much from these outbursts except locally, returning to their normal course after passing beyond their influence.

One of the results of trap-intrusion is to convert the talcose schists into a coarse steatite, or potstone, used in the manufacture of domestic ware (plates and basins), which is still made for the Calcutta market.

Quartz-pebbles strew the ground in all directions, being the result of weathering. Quartz is plentifully distributed throughout the schists in strings, and lenticular and segregated masses (especially the latter) in the neighbourhood of trap-eruptions. But the quartz is never found in any continuous form (such as reefs).

Where the amount of upheaval is small, the copper-belt is covered by alluvium, and undoubtedly exists regular and well-defined under this covering.

The other metals found along this belt are iron and gold. The former is in the shape of hæmatite and magnetite. The hæmatite is associated with the schists, and found as a filling in the vein-stuff; the magnetite is found as thin beds in quartzite. The hæmatite is used extensively by the native iron-smelters, and the iron is of excellent quality, being almost equal to the very best Swedish.

Gold is found in the rivers and watercourses after the rainy season, in the shape of wiry and flattened pieces. It is, no doubt, distributed in sparse quantity throughout the schistose zone. After the weathering of the soft rocks during the hot season, many tons of *débris* are every year carried down the rivers during the rainy season, and subjected to the churning action of water, the lighter gangue being carried away, and the heavier gold lodged in the nooks and crannies of obstructing rocks in the river-beds.

The precious metal is sought after by a class of low-caste natives called Jhoras (Fig. 7, Plate XXIV.); this occupation being considered a degrading one by the better-class natives. The Jhoras scatter themselves throughout the country, each party having its favourite locality. The yield sometimes is only one anna per day per person, a production of four annas being considered a red-letter day. The tools used are a wooden dish (Figs. 8 and 9) and a scraper (Fig. 10, Plate XXIV.). The gold, which is mostly in the form of dust, is melted into a button, and sold to the sonar or goldsmith, of which nearly every village has its representative.

The copper ore occurs disseminated, and in beds or segregated masses, throughout the whole belt, along the lines of least resistance, and following the dip and strike of the country, so that the ore partakes of the nature of a stratified mineral deposit. At the time of the formation of the copper-belt, water charged with copper in solution would become precipitated in the form of strings and bunches along the planes of foliation. Where the solution has been concentrated the deposits would be larger. After the building up of the beds and tranquil deposition of mineral matter, the whole belt has been upheaved, when a further concentration would take place. The great number of slickensides points to the movement of the beds at different periods; and lastly, the intrusions of trap would still further enrich the ore-beds in their vicinity. Practical search has shown that the ore-beds improve and become more defined as depth is attained, clearly indicating that the copper deposits in this district are likely to be found permanent.

Ancient Mines.*

Wherever the force of upheaval has formed outcrops, innumerable old workings are found, evidently the work of a once industrious people. According to tradition, Ruam, probably a king of Orissa, invaded this district some 1,000 years ago, and took up his abode on one of the spurs of the Sideshur hill, called Ruamgurgh, and fortified it. Extensive old workings, heaps of slag, and remains of walls at the present day testify to the story. This king is reputed by the people of the district (evidently Santals) to have been a foreigner, and to have been conversant with two languages, probably Bengali and Oorya.

In addition to the great number of old workings visible at the present day, there must be a good many more filled up with *débris*, and hidden by jungle-growth.

In some places, particularly Landu and Rakka, the old workings are so plentiful that the resemblance to an immense rabbit-warren is very noticeable.

Where the copper-bed is parallel with the side of the hill, and within easy distance of the surface, small circular shafts have been sunk some 20 feet apart, all down the hill to the water-level of the country, and communications made from one to another. No obstacle appears to have deterred these people from working the ore, and bunches of ore have been worked in the quartzites on the top of Rangamatia on its precipitous side.

In every instance the softer portions have been worked, leaving in some cases pillars of less rich material as supports to the roof, where the deposit has been extensive, like that at Chundra hill.

Close by the old workings are heaps of slag, showing that the ore was converted into metal on the spot. The bulk of the ore has occurred as laminations in the schists and quartzites, so that a rough concentration would increase the amount of slag, and probably much of the slag results from iron smelting.

There are no indications of a sulphide ore having been worked, and either the existence of large quantities of easily smelted ore or ignorance of the method necessary to deal with sulphide ores would lead to work being abandoned on approaching copper pyrites.

The amount of work done over so considerable an area indicates that the extraction of ore must have been carried on for a considerable number of years.

* Interesting information will be found in the *Records of the Geological Survey of India*, vol. iii., page 86.

Singhbhoom Copper Company.

Attention was called to this district in 1857, and the initiation of the Singhbhoom Copper Company must have been rather a bold undertaking in those days, when railways were in their infancy, and the climate by no means a healthy one, because at that time there must have been extensive forests and thick jungle extending in all directions. The route taken would be by road to Midnapore, and thence by mere cart-tracks across the country. The distance from Midnapore to the head-quarters of the company at Landu would be 86 miles.

Operations appear to have been conducted over a length of 16 miles of country, through Jamjhora, Landu, Hitku, Matku, and Rajdoha. Many shafts, adits, and inclines scattered throughout this area point to the vigorous search for workable deposits, but these explorations were generally confined to the outcrops, where much of the ore must long previously have been extracted by the ancients. The main working was at Landu on the middle bed (Fig. 11, Plate XXV.).

The natural result of so extensive an exploration was the expenditure of a large sum of money, but the culminating point was reached, when a smelting-works was insisted upon before the mines were in a sufficient state of development. Rajdoha was selected as the site for the smelting-works, evidently on account of its water-supply (the river Gurha runs by the works) and a building was erected 100 feet long by 20 feet wide, containing furnaces on the German-hearth principle, and alongside this building another was placed for the housing of the boiler and engine. The boiler is about 16 feet long with flattened ends and made of plates $\frac{1}{4}$ inch in thickness, working possibly at a pressure of about 10 lbs. to the square inch. The existence of a ring, roughly riveted round the centre of the boiler, points to the riveting having been done at the mine, and to the boiler having been transported to the mine in two pieces.

The engine, which was seriously damaged when the writer saw it, is an old-fashioned beam-engine, all the parts being of chilled cast-iron, fitted with condenser and fly-wheel and a pump to feed the boiler. This engine supplied the blast to the furnaces. The company had agreed to pay the yearly royalty of rs. 9,200 (say, £920 in those days) to the Rajahs of Seraikela and Dhalbhoom, for the right to mine, and, their resources being exhausted, all work was abandoned in 1859.

Hindostan Copper Company.

In 1862, work was commenced, and from the position of the buildings, Rajdoha appears to have been the head-quarters, for the officials of the company.

Some copper was smelted and sent to Calcutta, *via* Purulia, to Raneegunge on the East Indian railway, a distance of 120 miles, so that this route would supersede the one (*via* Midnapore) of the first company. The smelted ore was all that could be desired.

The only evidence of mining by this company is the existence of an underlie shaft at Rajdoha, sunk to a depth of 100 feet, with levels at the depth of 60 feet, carried in 9 and 10 feet respectively. The ends show impregnations and laminations of carbonates and oxides, but nothing of a solid nature. To the north of this underlie shaft a vertical shaft was carried, sunk to a depth of 70 feet, with the evident intention of meeting with the underlie shaft at a lower level. This vertical shaft had just struck pyritic ore when work was abandoned. Although the annual rent was reduced to £450 it was not wholly paid, and on the suspension of this company, in 1864, the brasswork of the engine and the doors and windows of the buildings were removed by the landlord.

Rajdoha Mining Company.

Early in 1891, on the opening of the Bengal-Nagpore railway, attention was once more directed to this district, owing to its proximity to the new line of railway. The most important portion of the copper-range in Dhalbhoom was leased direct from the Government (the estate having sometime previously come under the Encumbered Estates Act). The area of the concession was 120 square miles, commencing at a point 16 miles from Kandra station on the Bengal-Nagpore railway, where the estates of Seraikela and Dhalbhoom meet, and extending in a south-easterly direction for a distance of 40 miles, with an average width of 3 miles. The concession included the sole right of prospecting over this tract of country for a period of five years, during which time selected areas might be taken up for mining on leases—for gold for a term of thirty years and for other minerals for forty-five years, with a yearly rental of rs. $3\frac{1}{2}$ per acre for land taken for gold, and rs. $1\frac{1}{2}$ (the present rupee is worth 1s. $1\frac{1}{4}$d. but is constantly fluctuating) per acre for land taken for other minerals. The concessionaires have the option of abandoning any land taken up under lease at any time, on payment of one year's rent, the minimum rent being rs. 500. The royalty payable is 5 per

cent. on the net profits; and on transfer of land to others a commission of 2½ per cent. The bed of the Subanrekha river, which flows almost the full length of the concession, is included in the area taken up.

Out of this district of 120 square miles the Rajdoha Mining Company selected an area, including the buildings and smelting-house at Rajdoha, which became the head-quarters of a company for a second time. The area of this property is 24 square miles, and extends from Rajdoha to Rakka in a south-easterly direction. The nearest point to the new railway is 24 miles, and the farthest point 32 miles.

Work was commenced in April, 1891, by the writer, and the buildings put into order. He decided to abandon the ancient workings, leaving a barrier of sufficient thickness between the old and new workings, and to sink two shafts, so as to prove an extensive area of the formation. After carefully examining the estate, two sites were selected at the extreme ends of the estate, viz., Rajdoha and Rakka, thus dividing the property into two sections (Fig. 12, Plate XXV.).

The shaft at Rajdoha, owing to its position in reference to the old workings, required to be carried down to a vertical depth of 350 feet to meet with the main bed (worked by the ancients). The shaft at Rakka, owing to a more favourable angle of the dip, and being placed at a nearer point to the old workings, necessitated a depth of only 203 feet to prove the same deposit.

Ultimately efforts were concentrated on the shallower shaft at Rakka.

The amount of work done at Rajdoha was as follows:—A circular shaft was commenced 17 feet in diameter, and reduced to 14 feet at a depth of 56 feet, where the solid ground was reached. This portion was bricked to a height of 6 feet above the surface, so as to allow of the tipping of *débris*. At a depth of 30 feet water was first met with, and became a serious obstacle to the further sinking, owing to the want of winding machinery. To remove the water and sinking-stuff, women were employed in eight hour shifts until winding-engines were placed east and west of the shaft. At a depth of 48 feet, copper pyrites was found disseminated throughout the country-rock (a slaty-blue schist). Copper pyrites in spots, strings, and bunches continued down to a depth of 130 feet, large bunches and strings of copper pyrites being found thickly mixed with the schists, and especially in segregated masses of quartz. At this point work was suspended.

The shaft at Rakka was commenced of the same dimensions as the one at Rajdoha, and after reaching a depth of 51 feet was reduced to 14 feet in diameter and bricked to the surface. The first signs of ore

(sulphide) were met with at a depth of 21 feet, and from this point to a depth of 76 feet, when a cross-cut was driven a few feet to the north, there was a thickly disseminated mass of ore-ground, assaying about 1 per cent. of copper. Every foot below this depth carried ore in strings, spots, and bunches, until at the 138 feet level, another cross-cut was driven a few feet to the south, and sinking resumed. At 158 feet, a two-feet bed of copper-ore was passed through, for the most part in quartz (this bed out-cropping in the gorge west of the shaft). This bed was resting on a highly indurated bed of quartzite, 10 feet in thickness, and spangled with copper pyrites throughout. The hardness of this material severely tried the native miners, and they were only able to drill 3 inches in 24 hours; but later there was no difficulty in removing the rock, electric blasting being used after the shaft had got beyond 100 feet in depth. Immediately below the bedded quartzites came a thick stringer of copper pyrites 4 to 5 inches thick. After this the usual formation was passed through, viz., chloritic schists with strings and veins of quartz, with the usual quantity of copper pyrites disseminated throughout. At 204 feet, a solid stringer of ore was again met with 4 inches thick. At 208 feet, or within 5 feet of the original estimated depth, the main bed was successfully proved, averaging 3 feet in thickness. It consisted of a mass of chloritic schists and quartz heavily charged with copper pyrites, the central portion containing a band of almost solid copper pyrites 8 inches in thickness (sometimes increasing to 18 inches in the levels). Sinking was continued below this point to a depth of 248 feet. At 233 feet, cross-cuts were driven north and south from the shaft. The ground below the main bed was of the same nature as the original mass, but the last few feet passed into bedded quartzites. The ground below the 233 feet level formed a sump 15 feet deep (Figs. 13, 14, and 15, Plate XXV.).

Further sinking was then suspended, and work mainly confined to the 138 and 233 feet levels.

The south cross-cut at the 138 feet level was driven 25 feet, and intersected the two-feet bed, and further extended to meet the main bed at a higher level. This, however, only reached 51 feet, leaving another 51 feet to meet the main bed, when work was stopped. Levels were opened out on the two-feet bed, east and west, 11 and 12 feet respectively and were stopped in ore.

After successfully proving the main bed by a cross-cut driven north at the 233 feet level, it was continued for 88 feet, and was about to intersect the two-feet bed at a lower level, when work stopped. The ground passed through resembled that met with in the sinking.

Levels were opened out on the main bed, east and west, 67 and 66 feet long respectively, and off these levels two winzes were sunk 30 and 27 feet deep, all in ore when operations ceased for want of funds.

To further prove the formation to the deep, a cross-cut was driven south for a length of 177 feet, and passing through the bedded quartzites, continued in chlorite-schists and quartz-veins with bunches, strings, and spots of copper pyrites scattered through every foot, the only exception being the appearance of two bands of magnetite (3 inches thick) intermixed with a few spots of copper pyrites.

Deep sinking had thus successfully proved the existence of two workable ore-bodies, together with a copper-permeated formation, 359 feet in thickness (ore was first met with at a depth of 21 feet), with excellent indications to the dip of a most promising character, clearly showing a possibility of other concentrated beds. The ore was in every case of a somewhat low grade, but in abundant quantity, and this implies the necessity of mining on a large scale.

The development and sinking were performed by winch-engines placed east and west of the shaft, while on the south side a winding-engine (with two cylinders, each 14½ inches in diameter) was in course of erection. Five boilers (Lancashire and Cornish) of about 20 horse-power each were already in position, together with a Schram air-compressor sufficient for eight drills (four drills were obtained) erected and housed.

The following assays show the value of the copper-ore:—

Two-feet bed. 245 bags (10 tons) of copper ore shipped from Calcutta in 1892:—(1) Copper, dry assay, 4⅛ per cent.; wet assay, 5½ per cent.; sulphur, from 5 to 10 per cent.; silver, about 3 ounces per ton; and gold, *nil*. (2) Copper, 5·3 per cent.

Picked ore from main bed. 249 bags (10 tons) of copper ore shipped from Calcutta in 1892:—Copper, dry assay, 13⅜ per cent. 1,110 bags (60 tons) of copper ore shipped from Calcutta in 1893:—Silver, traces; gold, *nil*; copper, dry assay, 13½ per cent.

Sample of thick stringer of copper ore from 204 feet level:—Copper, wet assay, 29·32 per cent.

A sample of copper ore found in 233 feet level south cross-cut was analysed as follows:—Copper, 11·20; cobalt, 0·39; iron, 30·45; sulphur, 19·77; silica, 24·45; alumina, 2·50; magnesia, 4·24; oxygen, water, alkali, gold, and silver, 7·00.

Picked copper ore from the main bed had the following composition:—Copper, 15·50; iron, 26·25; silica, 21·00; sulphur, 21·50; alumina, 2·60; phosphorus, 3·25; lime, 3·50; magnesia, 2·00; nickel and cobalt, 1·20; silver, 0·004; water, 1·35; total, 98·154.

The absence of arsenic in all of the assays is a most favourable feature (as arsenic tends to make copper brittle and interferes with the successful production of sound metal).

Gold.—Two experiments were made with a small sluice in the river Gurha, about 4 miles east of Rajdoha, with the following results:—From December 9th to 14th, 1891, or six days sluicing, yielded only 9 grains, of which one small piece weighed 1 grain. The next week's yield was:—

		Grains.
December 16th	yield	¾
17th	„	½
18th	„	1¾
19th	„	1¼
20th	„	1
21st	„	1½
Total ...	„	6¼

The amount of alluvial dealt with was a ton or thereabouts per day, so that this produce represented an average yield of about 1 grain per ton. Some of the float-gold was lost.

The above result at any rate precludes any attempt to carry on successful alluvial gold-mining in this area. But there is a possibility that gold may occur in the quartz-reefs which are supposed to exist in the gneissose area to the south of the Sub-Metamorphic rocks.

Labour.—The labourers consisted for the most part of Santals (aboriginees) and Bengalis. They are quick to learn, and would become a very useful class of miners if they would only attend regularly at work. Native festivals, which are of almost weekly occurrence, are the curse of the country. On these occasions the workmen leave without the slightest notice, regardless of the well-being of their employers. From this cause progress is slow, and the patience of the management often completely exhausted. It will be a fortunate day when the Government steps in and affords the mine-owners protection from such conduct. Regular attendance will not only benefit the country, and increase its production, but will be an incalculable boon to the native, and enable him to indulge in many luxuries that this hand-to-mouth policy prevents, and keep him out of the hands of the money-lender who is always on the alert to secure his house and chattels.

A great deal of patience and firmness was necessary to train raw villagers to the use of the hammer and drill.

The indurated character of the formation was a bar to speedy work, so that four men per shift of eight hours would only advance the face some 3 inches. The rate per week did not exceed 4 feet, and generally did not reach that. A pair of Cornish miners would probably drive 12 feet, or three times as much; but, on the other hand, an European would get twelve times the pay.

Wages.—

					Annas.
Underground, sinkers	per shift of	eight	hours	...	4
„ miners	„	„	„	...	3
Surface, landers	„	„	„	...	2½
„ males	„	ten	„	...	2
„ females	„	„	„	...	1
„ boys	„	„	„	...	1
„ girls	„	„	„	...	¾

			Rupees.
„	engine-drivers	per month ...	8 to 12
„	carpenters	„ „ ...	12 to 20
„	blacksmiths	„ „ ...	8 to 20
„	strikers	„ „ ...	4 to 5
„	fitters	„ „ ...	20 to 30

Fuel.—Small coal costs about 13 rupees per ton delivered at the mine. Small vertical boilers with cross tubes were used, fired with wood-fuel. There is an abundance of scrub jungle all over the estate, and this was used with great benefit during the dry season, but in the rains great difficulty was found in keeping up steam. Only the poorer kinds of wood were used. A bundle 2 feet long and 1 foot in diameter costs ½ anna, 1 pice (4 pice to an anna) going to the estate as royalty, and 1 pice to the woodman.

Timber.—Sâl and Ásan are to be obtained in the neighbourhood for timbering underground, but fortunately none was required, the rock being excellent in this respect. However, timber would be required eventually, should stopeing be resorted to. The estate charged 1 anna per inch in diameter at the thinnest end, irrespective of length, and the cost for transport would depend on distance and accessibility.

Building-materials.—There is fairly good clay for brick-making, and an abundance of concretionary limestone called kunkur, which is collected by the natives. Bricks, burnt with coal, cost 10 rupees per thousand; and limestone costs ½ anna per 80 pounds.

Transport.—There is a road 32 miles long to Kandra, of which 24 miles is not bridged. Bullock-carts carrying up to ½ ton are plentiful, and the cost is 4 annas for 80 pounds conveyed 32 miles. Heavy machinery must be pulled through the country by men, there not being any cattle fit for this work. The carriage of a 7 tons boiler costs 180 rupees, or say, £10 for the whole distance, but would be much less on a bridged road.

Climate.—This climate is very trying, especially to unacclimatized Europeans and natives, but not so much as Calcutta, although the latter city is more free from fever. The temperature is higher than that at Calcutta, but very much drier, and easier to bear. The temperature ranges from 50 degs. to 110 degs. Fahr. in the shade. The winter months, which are exceedingly pleasant, (were it not for the fever which lurks about after the rains owing to the abundance of jungle-growth), commence in November and terminates at the end of February. The hot season lasts from March until about June 10th, when the rainy season sets in, and continues until September. When the district gradually gets cleared of jungle, in the neighbourhood of settlements, a healthier state of things will ensue.

Rainfall.—A rain-gauge was introduced in June, 1891, and the following records have been taken:—

1891	Inches.
June 27th to 30th ...	1·94
July	12·69
August	17·41
September	13·25

1891	Inches.
October	0·88
November	1.00
December	Nil
Total	47·17

1892.	Inches.
January	Nil
February	1·06
March	Nil
April	0·20
May	0·52
June	7·94
July	19·01
August	4·58
September	10·71
October	2·97
November	5·12
December	Nil
Total	52·11

1893.	Inches.
January	0·29
February	6·23
March	2·13
April	0·55
May	8·57
June	16·38
July	13·45
August	11·16
September	10·93
October	5·32
November	0·11
December	Nil
Total	75·12

1894.						Inches.
January	...	...	...	...	...	Nil
February	...	...	...	...	...	0·55
March	...	...	...	...	...	Nil
Total	...	...	...	...		0·55

The year 1893 was an abnormally wet season, such as had not occurred for a great number of years. The year 1892 was a normal rainy season for the Chotà-Nagpore district.

Remarks.—When the East Coast railway, which is now in course of construction, reaches Midnapore, it is proposed that the Bengal-Nagpore Railway Company should extend their line from Sini to Midnapore. When this scheme is carried out, there will be railway communication along nearly the whole length of the copper-belt, and the Rajdoha Mining Company will be within 1 to 2 miles of the railway. This railway will undoubtedly give a fresh impetus to mining enterprise, and there is no reason why Rajdoha should not gradually become a busy mining centre. There is abundance of room for innumerable mines. The alignment of the proposed railway has been surveyed three times, and does not present any great difficulties except the bridging of the Subanrekha river.*

Baragunda Copper Mine.

This copper-mine is situated 24 miles south-west of the Giridih station of the East Indian railway, in the Palgunj estate, in the Hazareebagh district (86 degs. 7 mins. east and 24 degs. 5 mins. north). Giridih is the terminus of a branch line, and serves the East Indian railway, and the Bengal, and Equitable coal companies, which are working mines in the Kurhurbari coal-field ; it is about 203 miles from Calcutta.

An excellent Government road runs from Giridih to within 5 miles of the copper-mine, at which point the company's road leads to the mine. The only obstruction on the Government road is the river Barakur, which is not bridged, and here the copper company have an aerial ropeway to convey the ore to Giridih to be smelted and to convey coal to the mine, during the rainy season.

Geology.—This deposit, which is of small extent in comparison to that of Singhbhoom, is of a similar nature. A belt of schistose or Sub-Metamorphic rocks have an east-and-west strike through the country, associated with occasional beds of quartzite, the whole resting on a great

* It is stated that the construction of this railway has been sanctioned by the Secretary of State.

area of gneiss: it has been upheaved from the south to almost a vertical position. The force of upheaval has been much stronger than in Singhbhoom, and Parasnath, a mountain 4,500 feet high, is the result of this great disturbance, being composed entirely of contorted gneiss.

The ore occurs disseminated through the schists, but the rock is almost all mica-schist, which is very friable and easily worked. Unlike the Singhbhoom copper-deposits, it is full of garnets of all sizes, especially near the surface, and nodules of lead and samples of zinc-blende are often found.

The bed consists of a low-grade mass of copper pyrites, the back of which has been removed by the ancients, and must have contained the same ore in a richer condition. An average assay of the bulk would give 1 to 1½ per cent. of copper.

This is undoubtedly a stratified mineral deposit, the ore occurring in strings and bunches along the planes of foliation. The bed, which averages about 14 feet, varies from 7 feet to 22 feet in thickness. It has the appearance of several beds, but non-productive masses occur, like horses, through the deposit, splitting up the bed, which ultimately meet again.

The bed-matter is full of slickensides pointing to the movements which have taken place at different periods, and these occurring in such soft rock increase the danger of working.

This deposit, which has now the form of two enormous old opencast workings, has been extensively worked by the ancients, probably at the same time as the Singhbhoom mines. Only the back of the deposit has been worked, or the decomposed copper pyrites, which, from the soft nature of the containing rock, would become influenced by sub-aerial agency to a great depth.

The ore was transported by these people to the village of Jereedee some 2 miles east of the mine, and here immense heaps of slag testify to the amount of work accomplished (Fig. 16, Plate XXV.).

The opencast workings are respectively 1,400 and 1,000 feet in length, with walls on each side, and 120 feet in width at some points. Between the two basins is a mass of unproductive matter which has never been examined in depth.

According to the name of the place, it would appear that twelve mines existed here, because *bara* means twelve, and *gunda* or *khunda* pits or workings, but at the present day the character of the place has changed, which is no wonder, after such a lapse of years, and the only remains are the two large opencast basins.

The ancients had reached a depth of 120 feet, earthen pots having been found at that depth, evidently used for the removal of water. There does not appear to have been any attempt at underground mining, and the ore-ground would be probably removed in steps.

After the formation of the Baragunda Copper Company in 1882, five shafts were gradually opened out on the back of the deposit, four in the larger and one in the smaller basin. The western shaft became the deepest point, reaching some 330 feet in depth. From this five levels were opened out, and gradually communicated with the other two shafts to the east (Fig. 17, Plate XXIV.). A large amount of stopeing ground was laid open, but the shafts were so small, and the appliances so inadequate to deal with a large quantity of low-grade ore, that the mine in some ten years never produced above 40 tons of copper ingots per month, and the average for the best years was probably not more than 25 tons per month.

There was no question of a large continuous supply of ore, because when work ceased the lowest points gave no signs of diminishing either laterally or vertically. In all probability the first evidence of diminution in the ore-bed would appear laterally, as the deposit partakes more of the nature of a basin. Success then depended on the mining of a large quantity of ore, and the only means of gaining this end would be by laying out the workings on a modern system.

In the first place, an error was committed in opening out shafts in the deposit itself, which necessitated constant timbering, and an uncertainty as to their permanency in such dangerous ground. The shafts were so small that there was only room for one kibble to work; and what with the want of guides, and the shafts not being perpendicular, the kibble was constantly striking from one side to the other, much to the detriment of the timbered sides.

In so vast an extent of workings the only pumping appliance was a bottom set, consisting of a bucket-pump 4 inches in diameter and 36 inches stroke, devoid of clack-doors, so that the column had to be broken each time the valves got out of order. The valve was below the sump-level and necessitated all repairs being done under water. As the top lift consisted of a plunger, 6 inches in diameter and 36 inches stroke, a large quantity of the water was continuously pumped over again.

When the shafts should have been free to raise mineral they were, especially during the rains, occupied in winding water. The immense basins formed a large catchment area for the heavy rainfall; and large bodies of water, pouring through the loose ground into the old workings, inundated the mine and greatly retarded mining operations.

In such dangerous ground with wide stopes and innumerable slicken sides, one can scarcely credit that overhand stopeing would be attempted with such unskilled labour as that of the native miner. Hundreds of tons fell away and filled the lower levels. Timber, which was not of the best, but the only material available, was used to keep the stopes open, the stopes being as much as 22 feet wide at different points, whereas judicious filling would have made the mine much more secure and more cheaply worked in the long run. The ore, when broken, was handled three times. After being automatically filled into waggons below the stopes, it was unloaded at the mouth of the level and loaded into kibbles, unloaded at the surface into waggons, thence unloaded into ore-bins, and loaded again, and unloaded at the mills.

If the shafts (two would have been ample) had been sunk, off the ore-body, in compact ground, timber could have been dispensed with, and any weak places in the shaft protected with brickwork. The shafts would have been large enough to enable adequate pumping plant to be provided; and with large cages fitted with guides, the ore could have been loaded at the stopes, raised to the surface, and run direct to the mills. Access from the shaft to the ore-bed, would have been obtained by cross-cutting; in the event of the stopes giving trouble, the shafts would not have been affected; and there would have been no serious difficulty in keeping the workings clear of water.

By the introduction of underhand stopeing, thus securing the proper protection of the miners, and judicious filling of abandoned stopes, the mine would have been working at the present day, and providing fairly good dividends. If mining be ever attempted again, that system, or something of a similar kind, will have to be introduced to ensure success, and a safe barrier of untouched ground should be left between the old and new mines.

Mills.—To deal with the output of the mine, five dressing-works were provided, consisting of twenty-one jiggers (three French), together with the necessary stone-breakers, slime-separators, and eleven buddles, capable of producing about 20 tons of dressed ore in a day. A small quantity of picked ore was also available. The ore was dressed up to 6 per cent. (dry assay), and this would represent sufficient production for 4 tons of copper ingots per day, or, say, 100 tons of metal per month, which should have been the output of a mine of this description. Unfortunately the production was only about 1 ton per day, consequently the mills were never fully employed.

Labour.—This was plentiful, well trained, but lazy, because the miners were paid a daily wage, and not in proportion to the work done.

In so soft a gangue, jumpers would have had more effect than hammers and drills, and the drilling would have been speedier, with half the amount of labour. The timber-men, who were invariably Santals, were excellent men, and considering the dangerous nature of their employment—fixing heavy timber in unprotected overhand stopes—performed remarkably good work.

Fuel.—The proximity to the coal-mines made fuel comparatively cheap, and on this account it would have been wiser, with such low concentration, to have erected smelting-works at the mine, and saved the cartage of so large a quantity of low-dressed ore. The erection of a costly aerial rope-way across the Barakur river, which was only used some four months in the year, would have been avoided, as a stock of coal might have been stored at the mine, sufficient for both mine and smelting-works during the rainy season.

Remarks.—The climate is more healthy than Singhbhoom, although Europeans occasionally suffer slightly from fever. The sanatorium, Parasnath, 4,500 feet high, is within 9 miles of the mine, and can be resorted to in event of sickness.

The rainfall and temperature are about the same as at Singhbhoom, but the atmosphere is drier.

The writer was in charge of this mine during 1890, and introduced a few necessary reforms. The mine was closed in May, 1891, owing to want of funds, about six months after the writer had left.

Palgunj Gold-prospecting Syndicate.

The estate of Palgunj, of which Baragunda formed a small part, can be conveniently approached at all points from Giridih. The total area is 400 square miles, and extends about 20 miles south of the Barakur river, and from Narungo about 20 miles east along the southern bank of the same river. The writer prospected over about half of this estate during three months of 1891.

Geology.—The formation on the south and north consists of bedded gneiss. It is occasionally cut up by segregated and vein-like masses of quartz, small hills of basalt and trap-dykes. The centre, in which the Baragunda copper-mine occurs, is a wide belt of schistose rocks, which extend east and west throughout the Palgunj estate.

The whole of this area has been violently upheaved from the south, so that the gneiss and schistose rocks are in a perpendicular position. One of the results of this severe disturbance is Parasnath hill, which is composed entirely of contorted gneiss. North of the schistose belt the gneiss has been elevated from the west, and has an inclination of 26 degs. to the east. This dip of the gneiss in this northern division can be traced for several miles east and west of the Barakur river.

The metalliferous deposits over this area have in every case the character of beds, and lie along the bedding of the rocks, the whole being evidently the result of tranquil deposition during the building up of the formation, and before the upheaval and intrusive action of igneous matter.

In some localities the gneiss looks like pegmatite, while in other localities it takes the form of graphic granite, where the arrangement of the crystals is transverse to the laminæ. The gneiss in depth alters to a granitoid nature.

Geyjhadah Old Workings.—These are situated in a small hill running almost parallel with and about 2 miles south-west of Baragunda. The formation is in every respect identical with that at Baragunda. There appear to be four beds, on three of which old workings are visible, in the shape of small basins, which have become silted up. The principal workings are in the No. 3 bed (Fig. 18, Plate XXV.).

Slag is scattered about the neighbourhood, and evidently carbonate and oxide of copper were extracted, as impregnations of carbonate can be seen amongst the *débris* of the old workings. Three small prospecting shafts were put down in the most favourable places, and encrustations of peacock ore and bunches and spots of copper pyrites were obtained. The work only lasted some two months, with the following results:—

Shaft on No. 1 bed reached 24 feet.
" " 2 " " 36 "
" " 3 " " 36 "

The Baragunda copper belt was also examined for 6 miles in an easterly direction as far as the Government road, and several prospecting-shafts were sunk, the deepest reaching 32 feet, with indications of copper pyrites. The copper belt extends for some considerable distance beyond the Government road and into another estate.

In a westerly direction the belt was traced for 2 miles to the end of the estate, and into the Rampore property. Three small shafts were sunk in this direction, and several pieces of galena met with in the sinking at Jungeedeere.

The results of this limited prospecting were not encouraging, and it appeared doubtful whether workable deposits existed east and west of Baragunda; if they do, they must exist below the unproductive surface-rock.

No traces of gold were found.

Narungo Tin-mine.

This is situated in the extreme north of the Palgunj estate. It lies 5 miles north of Baragunda and 18 miles east of Giridih (86 degs. 7 mins. east and 24 degs. 10 mins. north). The Barakur river is close to the mine, and has to be crossed to reach Giridih, to which place there is a fairly good road.

Geology.—The geological aspect has already been described as being composed entirely of gneiss through which occasional veins of quartz, eruptive and intrusive matter occur. The beds lie at an angle of 26 degs. to the east, and the strike is north and south. The ore consists of stratified mineral deposits, as already mentioned.

Workings.—The working of tin does not appear to have been known by the natives of India, and the discovery of this deposit was a pure accident.

At the village of Narungo are immense slag-heaps, the result of iron-smelting, which must have been carried on for some considerable time, and is still continued by the Kols (aboriginees). (Fig. 19, Plate XXIV.) It appears that they one day smelted what they thought was ironstone, but on tapping the furnace, much to their astonishment a white metal flowed out, which they mistook for silver. This they carefully conveyed to Raneegunge, the nearest important town in those days. Here this metal was seen by Mr. Lord, and its real nature and value ascertained. This gentleman in 1865 obtained the right to mine from the Palgunj rajah at an annual payment of £200, and Mr. Louis Deveria was entrusted with the opening of the mine. It was found that the tin cropped out in the form of a bed, in the sides of a nullah (water-course) and appeared to dip with the gneiss. An open cutting was made for 90 feet, and the bed followed under the gneiss for another 25 feet. The bed then gave signs of pinching out, and further work was abandoned. The oxide of tin, which was of excellent quality and free from sulphur owing to atmospheric agencies, was easily reduced by the iron-smelters. It is understood that about ½ ton of tin was produced and sold. This

would have amply repaid all expenses and given an excellent profit if work had been further continued, with the same promising results.

In 1891, the writer included this locality in his investigations, and was ultimately able to prove the existence of tinstone over a considerable area. On his arrival at Narungo in January, 1891, he found the incline full of water and *débris*. About 120 feet east of the commencement of the incline is a shaft 15 feet in depth, and again 50 feet farther east is another shaft 20 feet deep, with an incline just commenced running towards this shaft from the north. The incline was cleaned out, and carefully examined, and the bed of tin (which had been partly worked) was found in the end of the incline (showing no signs of having cut out), running parallel with the bedding of the country-rock (gneiss). The bed was 6 inches in thickness, the tinstone, in a decomposed state, being scattered through the clay, and black mica with crystals of magnetite are scattered over the top and bottom. To further test the formation the bottom of the incline was taken up, and disclosed another and better bed 4 feet below the former one, and running regularly with the bedding of the gneiss. This bed varied from 12 to 18 inches in thickness, and yielded some excellent stones of tin oxide.

The 6 inches bed was now abandoned, and work confined to the new bed.

The shaft was carried down to a depth of 20 feet below the old workings of 1865, with the bed running as regular as a seam of coal.

The old shaft was continued in depth, and intersected the incline at 30 feet below the surface, and still further carried below the floor of the incline to prove the formation, and reached a depth of 48 feet. From the bottom of the shaft at the incline-level, a level was driven south from the shaft for a length of 50 feet, and the bed opened up the whole distance, averaging 15 inches in thickness. This was the amount of work done at this point when prospecting work ceased.

The country was vigorously prospected, and tin crystals and gneiss were found scattered over some 21 square miles, Narungo forming the north-western corner.

The gneiss on the top of the principal ore-bed is compact and regular, thus forming an excellent roof. The ore is mixed with softer gneiss, black mica, and small veins of felspar. Crystals of magnetite are scattered throughout the gangue. Below the bed there is a thin layer of clayey matter, like a flucan, and below this follows bedded gneiss at first soft, but gradually becoming harder in depth.

Work was resumed in May, 1892, with the intention of developing

the finds of tin made by the writer in the previous year. The old incline, afterwards called the No. 1 underlie shaft, was enlarged (Fig. 20, Plate XXV.).

In December, 1892, the mine had reached a depth of 277 feet, with the bed 3 feet thick. Three levels were started off the shaft, with the following results :—

North	90	feet level,	165	feet long,	bed	1½	feet thick.
„	150	„	109	„	„	1½	„
„	210	„	15	„	„	1½	„
South	90	„	137	„	„	1	„
„	150	„	111	„	„	1½	„
„	210	„	16	„	„	3	„

Winzes had been sunk, and connected the 90 and 150 feet levels.

A second shaft, called the No. 2 underlie, was started 425 feet to the south of No. 1 underlie shaft, and reached a depth of 136 feet, with the bed of tin 6 feet thick, indicating that the deposit rapidly thickened towards the south. At this point work was suspended because water became a difficulty, and necessitated pumping-machinery (Fig, 22, Plate XXV.).

The only trouble met with up to this date in the No. 1 underlie shaft was a bar of quartz 12 feet thick, but this had not disturbed the continuity of the bed, thus indicating that the quartz must have been intruded into the gneiss after the deposition of the tin-bed.

At the end of 1893, the No. 1 underlie shaft had been further sunk to a depth of 614 feet, other levels being opened out at the depths of 280, 400, 450, and 600 feet respectively, but on these little work was done. The bed continued down regularly to 450 feet, varying from 12 to 18 inches in thickness. At this point the gneiss became very hard, and of a granitoid nature, and the bed dwindled down to only 6 inches in thickness. The appearance of the ground gave signs of being in the vicinity of a fault.

Work was continued until 568 feet was reached, when the bed cut out altogether, and this state of affairs continued until reaching 614 feet, when work was abandoned in the sinking. At 568 feet, the sinking passed through a downthrow fault, so at the 600 feet level a cross-cut was driven a few feet to the west, when the bed was successfully recovered, but, as expected, in a very disordered condition. Nothing further was done, unfortunately, for want of funds. Had work been further prosecuted in this shaft, the bed might have gradually become more concentrated. Although the tin-bed held downward as regularly as a seam of coal, with excellent stones of tin at intervals, yet the whole was not sufficiently rich to warrant stamping.

Magnificent tinstone was obtained on passing through the bar of quartz, but did not hold for any great distance (Figs. 20 and 21, Plate XXV.).

The No. 2 underlie shaft was then reopened, and the same bar of quartz passed through at a higher level, but the tin-bed did not improve in its vicinity. The sinking was carried down to a depth of about 300 feet, the tin-bed running regular throughout, and averaging about 4 feet in thickness over the whole distance below the quartz. At this point all work was stopped for want of funds (Figs. 20 and 22, Plate XXV.).

Further prospecting proved tin-bearing ground over a much larger area than that discovered by the writer in 1891.

Machinery.—Stamps were provided, and buddles erected to deal with the tin oxide, which at the outcrop gave such excellent promise. They only worked a short time, with a yield of black tin equal to some 4 tons of white tin (which was never smelted).

Results.—Mining over an infinitesimal portion of this tin-bearing area had thus proved the existence of tin-beds extending to the deep, regular and well-defined, in most congenial ground, and under the most favourable conditions met with in mining. It will be exceedingly strange, then, if valuable deposits of workable tinstone are not found on further development to the deep. The estate, which is such a vast one, may be considered as virgin country, and will require most exhaustive search before being abandoned as an unproductive field.

Not only in the Palgunj estate is tin expected to be found, but also farther north in Burkutta and Leda, where outcrops of tin-beds are known to exist. It is most probable, also, that further search in an easterly direction from Narungo may disclose a tin-bearing formation along the south bank of the Barakur river, and across that river into the estate of Deopore, a total area, roughly speaking, of upwards of 200 square miles.

The climate is the same as that of Baragunda, but being on elevated country, well cleared of jungle, and a sandy subsoil, Europeans keep much better health here than at the copper-mines.

Labour is plentiful, for the miners flocked to Narungo when work ceased at Baragunda.

Coal is cheap, Narungo being 18 miles from the collieries.

Timber suitable for mining purposes can be procured in the neighbourhood.

The rainfall and temperature are the same, in every respect, as at Baragunda.

The following results of assays of tin-ores were obtained in 1891:— One cwt. of stream tin was procured by washing the sand in the bed of the Munwarah river, some 2,000 feet south-east of the mine. From this an average sample was taken of 10 lbs. and roughly smelted. Two lbs. of ore were also taken from the mine and smelted. The cwt. of stream tin, together with the two samples of white tin, were sent to Calcutta, and assayed by Messrs. Smith, Stanistreet, & Co., with the following results:—(1) The stream tin on assay produced 215·2 lbs. of tin per ton of ore. (2) The roughly smelted sample from the stream-ore weighed 12 ozs. 150 grains, and contained 73·437 per cent. of tin, equal to 135·5 lbs. of tin per ton of ore. (3) The sample smelted from the mine-ore assayed 99·037 per cent. of tin.

On March 15th, 1891, a rough experiment was made to ascertain the cost of winning, dressing, and smelting 1 ton of dirt taken from a full section of the level (20 cubic feet being taken as a ton).

One ton of dirt taken from the full section of the level took three days to obtain at the following cost:—

	Rs.	a.	p.
36 miners at 3 annas	6	12	0
36 carriers at 2½ annas	5	10	0
Dressing, smelting, oil, tools, and supervision...	2	0	0
Total cost per ton	14	6	0

This 2,240 lbs. of mine-dirt was thoroughly mixed, 80 lbs. was taken from it as a sample and dressed by hand, and produced 56 lbs. of rough dressed ore, which yielded 1·5 lbs. of white metal. This would equal 1·87 per cent. of white metal.

The full section of the level was 4 feet by 6 feet, out of which 4 feet by 1¼ feet was tin-bed, the remainder having grains of tin oxide disseminated throughout.

The following interesting experiments were made in February, 1892, by Mr. Samuel Gifford, of Bristol:—

"Samples of the vein were taken from various parts of No. 1 shaft and the gallery, the stone in each case being crushed, weighed, and roughly washed to clean and concentrate the ore in it. The residue was then re-weighed to obtain some idea of the proportion of tin oxide in the vein, with the following results:—

No of Sample.	Weight of Raw Vein Stuff.	Weight of Residue after Washing.	Visible Gangue and Iron Remaining.	Weight of Apparent Pure Tin Oxide.
	Lbs.	Lbs.	Lbs.	Lbs.
1	42	11·0	4·0	7·0
2	30	11·0	5·0	6·0
3	36	8·0	4·0	4·0
4	18	1·0	0·5	0·5
5	30	4·0	1·5	2·5
6	18	4·5	1·5	3·0
Totals...	174	39·5	16·5	23·0

"This table shows that the proportion of apparent tin oxide in the was about 13 per cent. or 291 lbs. per ton of dirt; but several of t samples were from the best points, notably Nos. 2 and 3, and elim ating these the results would show: vein-stuff 108 lbs. yielded net ore 13 lbs. equivalent to 12 per cent. or 268 lbs. per ton.

"To prove the purity of the apparent tin oxide, etc., the whole of residuum from these samples was smelted in a native iron-furnace; and ing the clean ore, as shown at 23 lbs., this should have given, say at 70 cent., some 16 lbs. of metal, but it yielded only 7 lbs. which would reduce proportion of actual tin oxide to the total of vein-stuff by nearly 56 cent. of the figures already given. At the latter part of the smelt operation the furnace got a little choked, and some loss probably ens from this. Another smelting was then tried with 18 lbs. of the l washed ore, the operation being carried through without the sligh hitch, both metal and slag running off perfectly clean. The m weighed 7 lbs, but if the ore had been pure tin oxide there should h been 12·6 lbs., and it may therefore be reasonably assumed that average of well-washed ore will yield about equal to 50 per cent. the general commercial class, the balance being a little ordinary gan with a large proportion of iron in various forms.

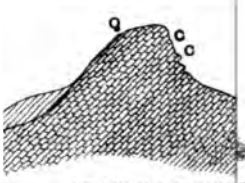
"It is very probable that, with proper dressing and concentrat machinery, the tin ores may be made as clean as the usual tin oxide with in commerce, but for present calculations it will be prudent to m the deduction noted.

"To further test the proportion of the ore to vein-stuff, other sam were selected and washed and weighed, as before, with the follow results :—

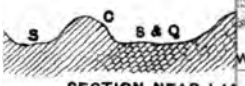
G. 2.

ON THROUGH RANGAMATI

SECTION NEAR RA



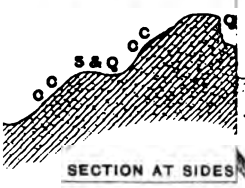
SECTION AT CHUN



SECTION NEAR LA

Fig. 7.

WASHERS OF SINGHBHOOM.



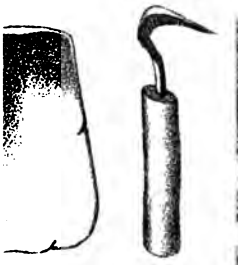
SECTION AT SIDES

Fig. 19.

OF PALGUNJ.

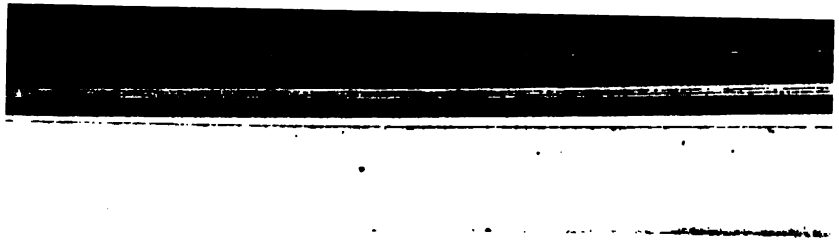
i. 9.

Fig. 10.



ING DISH AND SCRAPER

SINGHBHOOM.



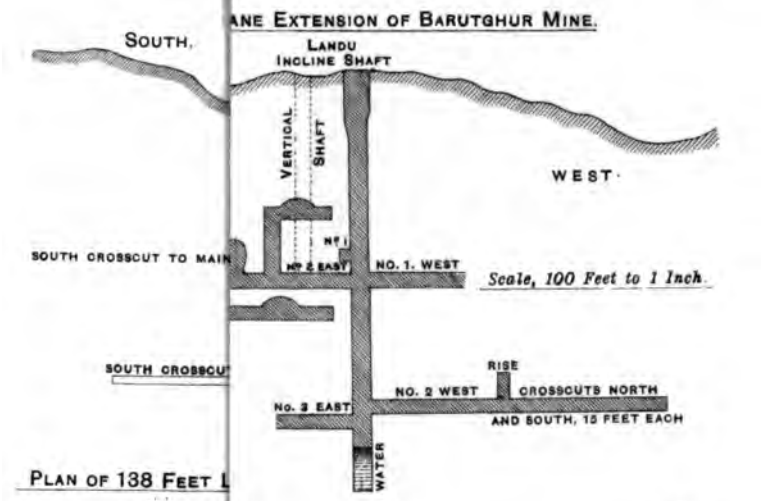

ANE EXTENSION OF BARUTGHUR MINE.
SOUTH.
LANDU INCLINE SHAFT
VERTICAL SHAFT
WEST.
SOUTH CROSSCUT TO MAIN
No 1
No 2 EAST
NO. 1. WEST
Scale, 100 Feet to 1 Inch.
SOUTH CROSSCU
RISE
No. 3 EAST
NO. 2 WEST
CROSSCUTS NORTH
AND SOUTH, 15 FEET EACH
WATER

FIG. 21.

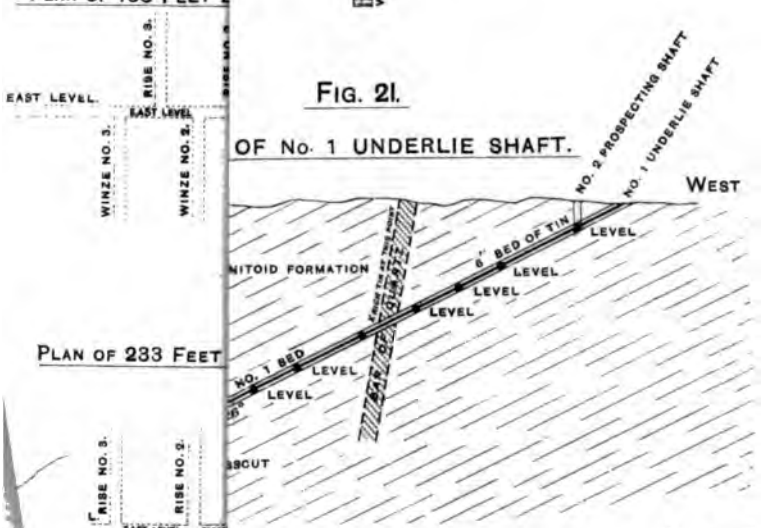
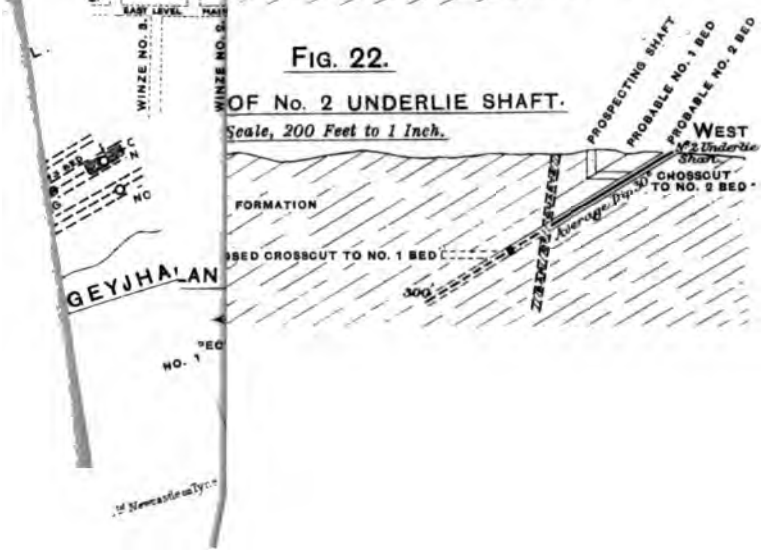
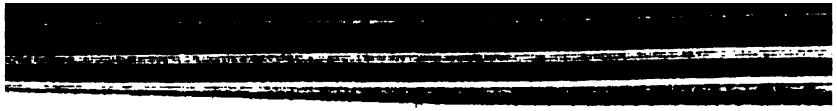

PLAN OF 138 FEET L
EAST LEVEL.
RISE NO. 3.
EAST LEVEL
WINZE NO. 3.
WINZE NO. 2.
OF No. 1 UNDERLIE SHAFT.
NO. 2 PROSPECTING SHAFT
NO. 1 UNDERLIE SHAFT
West
LEVEL
6' BED OF TIN
NITOID FORMATION
LEVEL
LEVEL
LEVEL
PLAN OF 233 FEET
NO. 1 BED
LEVEL
LEVEL
RISE NO. 3.
RISE NO. 2.
EAST LEVEL
WINZE NO. 3.
WINZE NO. 2.

FIG. 22.


OF No. 2 UNDERLIE SHAFT.
Scale, 200 Feet to 1 Inch.
PROSPECTING SHAFT
PROBABLE NO. 1 BED
PROBABLE NO. 2 BED
WEST
No 2 Underlie Shaft
CROSSCUT TO NO. 2 BED
Average Dip 30°
FORMATION
SED CROSSCUT TO NO. 1 BED
300'
GEYJHAIAN
NO. 1



No. of Sample.	Weight of Vein-Stuff.	Weight of Residue.	Deduction for Impurities as above.	Net Commercial Tin Oxide.	Proportion of Ore to Vein-Stuff.	
	Lbs.	Lbs.	Per Cent.	Lbs.	Per Cent.	Lbs. per Ton
1	6	0·50	50	0·250	4·0	
2	6	0·75	50	0·375	6·2	
3	6	0·25	50	0·125	2·1	
4	6	0·25	50	0·125	2·1	
Totals...	24	1·75	50	0·875	3·6	80

"From this table it would appear that something like 80 lbs. of commercial tin ore per ton of vein-stuff can be extracted along the gallery opened, and with an increased thickness of the lode this would be a very profitable yield."

The tin was assayed by Messrs. Smith, Stanistreet, & Co., and found to contain 99·85 per cent. of metallic tin. Lastly, a sample of tin oxide, met with in the tin-bed on the lower side of the bar of quartz in No. 1 underlie shaft, was submitted to Messrs. John Taylor & Sons, London, who reported that "The metal is of a very superior quality; the sample that we had assayed produced 73¾ per cent. of white tin. The produce and quality are much superior to those of the tin raised in Cornwall."

Conclusion.

In submitting this paper the writer trusts that he has, at least, succeeded in showing that in its metalliferous deposits Chotà-Nagpore possesses sources of wealth which may at no distant day add to the material prosperity of our great Empire in the East. He is much indebted to Messrs. Gillanders, Arbuthnot, & Co., of Calcutta, for placing at his disposal copies of assays and other information.

THE BLACKWALL TUNNEL.*

Fig. 1 (Plate XXVI.) is a general plan of the works, and Fig. 2 (Plate XXVI.) is a section, showing the lengths of the various kinds of work and gradients of approaches. The approach on the Middlesex side commences at the East India Dock road, opposite the entrance-gates to the docks. From this point, taking a southerly direction and passing under the Great Eastern and Midland Railways close to Poplar Station, the roadway descends by a gradient of 1 in 34·384, until the side of the river is reached, and it has attained the level fixed upon for the horizontal portion under the river. Crossing in a south-easterly direction from a point about 150 feet up-stream from Blackwall Stairs, the south bank of the river is gained, and here the gradient for the Kent approach of 1 in 36·194 commences, the surface-level in Blackwall Lane being reached opposite the South Metropolitan Company's gasholders.

Four large shafts of an internal diameter of 48 feet are placed in the positions shown in Fig. 2 (Plate XXVI.); these shafts are situated where changes of line or level occur, it being considered that a shield of considerable length, such as would be required for this tunnel, could not be made to work round curves. As the tunnel passes through these shafts, openings have to be made in the latter, in order to admit of the tunnel-lining. These openings are circular, and $29\frac{1}{3}$ feet in diameter.

It is intended to fix spiral staircases in Nos. 1, 2, and 4 shafts, so that foot-passengers living near these shafts may enter or leave the tunnel without having to go to the ends of the approaches to do so. No access to No. 3 shaft will be made from the surface, as it is covered in by a domed arch, which carries a ventilating-shaft.

Cast-iron Lining for Tunnel.—Two sections of cast-iron lining have been designed; the heavier section is to be used between shafts Nos. 1 and 4, for a length of 2,262 feet; the lighter section is intended to be erected between shaft No. 1 and the Great Eastern Railway on the Middlesex side, for a length of 821 feet, making a total of 3,083 feet of iron-lined tunnel.

The heavier section consists of a tube 2 inches in thickness, having flanges 12 inches deep, and from 2 to 3 inches thick; each ring is 2 feet 6 inches in length, and is built up in 14 segments, with a solid key at the top. The key is slightly narrower at the outside of the

* "The Blackwall Tunnel," *Engineering*, 1895, vol. lix., pages 1, 166, 206, 329, and 396.

lining than at the inside. This is to facilitate insertion when it is required to fix it, and the contiguous flanges differ from the others in being made to suit it. The weight of a complete ring is 16⅛ tons. All joints, both circumferential and longitudinal, are machined throughout, and recesses, 2 inches by ⅜ inch, are formed on the inside for caulking. The segments are bolted together with 1½ inches bolts of the various lengths required. Holes, 1½ inches in diameter, are drilled and tapped through the plates, through which grout is forced, by compressed air, to fill up any vacant spaces outside the lining. The holes are closed by wrought-iron screwed plugs when the grouting is completed (Fig. 4, Plate XXVI.).

The lighter section of cast-iron lining is a tube 1⅛ inches in thickness, with flanges 10 inches deep, and from 1½ to 2¼ inches thick; in all other respects it is similar to the heavier lining previously described. Its weight is 11⅜ tons per ring complete.

The outside diameter of both sections is 27 feet. It is intended to have an internal lining of special concrete 4 to 1, faced with glazed bricks.

Cut-and-Cover.—A light section of brickwork has been adopted for 436 feet of the cut-and-cover portion on the Middlesex side, where the depth of trench varies from 34 feet at High Street to 41 feet at the Great Eastern Railway (Fig. 2, Plate XXVI.) at which point the iron-lined tunnel begins, and for a length of 335 feet on the Kent side, the depth varying from 37 to 42 feet. From the latter depth, and until No. 4 shaft is reached, a heavier section of brickwork is used. The thickness of brickwork is respectively, for the two sections mentioned, 1 foot 6 inches and 1 foot 10½ inches; the inner ring, in each case above footpath-level, being of glazed bricks. Immediately outside the brickwork, a band of asphalte 1½ inches thick is placed to ensure water-tightness. The brickwork is backed up with 6 to 1 Portland-cement concrete built tight to the sides of the trench. A polished granite-moulding is placed at the end of the cut-and-cover arching on each side of the river.

Open Approaches.—The lengths of the open-approach structures on the Middlesex and Kent sides are 765 feet and 860 feet respectively. The structure is composed of 6 to 1 cement concrete, faced with brickwork above the footpath-level, and bonded to the concrete, the face being glazed, as in the cut-and-cover and tunnel. The face of the retaining-wall is battered 1 in 4½ up to surface-level, where a Portland stone string-course is placed, forming a base for the parapet-wall, which is finished

with a granite coping. A layer of 1½ inches of asphalte is carried through the invert and up the back of the retaining-walls. A staircase 6 feet wide is placed on each side of the road at the end of the open approach on both banks of the river.

The roadway will be 16 feet wide throughout, giving sufficient accommodation for two of the largest vehicles in use in London, side by side. There will be a footpath on each side of the roadway 5 feet 4½ inches wide in the open approach, and 3 feet 1½ inches wide in the cut-and-cover and iron-lined tunnel. The roadway is carried by a 9 inches brick arch, which forms the roof of a subway for pipes, etc., access to which is obtained by a shaft at the end of the cut-and-cover on each side of the river, thus obviating the necessity of breaking up the road in order to lay new pipes, etc., or for repairs.

Drainage.—Arrangements will be made for conveying all water collected in the open approaches on both sides of the river, and that used in cleansing the roadway, etc., to a well in No. 2 shaft, where permanent pumps will be erected to deal with it. The drainage of the tunnel is principally provided by means of pipes close to the kerbs. The water from these pipes is collected at the foot of No. 2 shaft and then pumped to the surface.

It is not anticipated that any special precautions will have to be observed in the matter of ventilation, but the entrance gauge-arches are 2 feet less in height than the tunnel, so that a ventilating-trunk can be inserted in the top of the tunnel. Illumination is to be by electric light.

Sinking the Caissons.—Arrangements were made to begin on the south side, thus driving northwards. The reason for this was that the ground was easier to work on the south side, and it was naturally concluded that the men would thus gradually gain experience in the various operations, and so be better able to deal with the exceptional difficulties which, it was known, would be met with later. It was therefore determined to commence the work by sinking shaft No. 4, which is the southernmost of the four vertical shafts that have been sunk for purposes of construction and affording access to the tunnel. The first two strakes are of steel, the rest being iron. The lower part of the caisson is shown in Fig. 3 (Plate XXVI.). The lower part of the caisson is strengthened by radial diaphragms, some of which extend upwards to a height of 48½ feet from the cutting-edge, and serve to strengthen the structure of the caisson, where its cylindrical form is interfered with by the tunnel-openings on either side.

Fig. 2 (Plate XXVI.) gives the positions of the vertical shafts. Each shaft consists of a wrought-iron caisson of 48 feet internal diameter, and of 58 feet external diameter at the bottom. The plates vary in thickness from $\frac{3}{4}$ inch at the cutting-edge to $\frac{5}{16}$ inch at the top, and are braced together with angle-irons. The plating is so arranged that the inner skin of the shaft is plumb, while for the outer skin the rings of plating are placed telescopically. That is to say, the inner skin plates are set at a slight angle from the vertical, so as to counteract the batter that would be given to the whole skin by the overlapping edges of the plates; while in the outer skin, on the other hand, the plates are vertical, and as they overlap there is a batter on the outside of about 1 in 100. This arrangement materially decreases the skin-friction in sinking. At a distance of $8\frac{1}{2}$ feet from the bottom, the inner skin is bell-mouthed to meet the outer skin, and thus forms a cutting-edge for the purposes of the sinking. This cutting-edge is strengthened by an external steel ring, 2 feet deep by 1 inch thick.

The space between the outer and inner skins of the caisson, varying from 5 feet at the bottom to about 4 feet at the top, is filled with cement concrete of 6 to 1; and in this way a solid cylindrical structure of great strength is formed.

The openings through which the tunnel-roadway will pass when the work is complete are closed during the operation of sinking a shaft. Iron plates forming a continuation of the outer skin, are bolted together and backed by horizontal and vertical girders bolted to the sides of the opening. The plates were bolted together instead of being riveted, so as to facilitate removal when the caisson was sunk to its full depth.

The iron lining of the tunnel and the inner-skin of the vertical shafts are bolted together; special tunnel-lining castings being used at these parts. Concrete (4 to 1) is used as packing between the cast-iron tunnel-lining and the projecting hood or collar. Dependence is not placed on the concrete to keep out water, but on the end-plate, the joints between the plate and the lining, and those between the plate and the collar being caulked with a mixture consisting of 1 part of sal-ammoniac and 448 parts of cast-iron borings.

The bottoms of the shafts are composed of concrete, and have a water-tight skin of iron $\frac{1}{4}$ inch thick placed near the bottom of the concrete. This arrangement is shown in Fig. 3 (Plate XXVI.), the two outer rings of plates being placed radially, and the central portion being filled in by square plates.

The method adopted for sinking the shafts was by excavation from

the interior, freeing the cutting-edge and building up the caisson as it sank down. As the operation proceeded, the concrete was filled in between the skins of the caisson, to give the required weight. The sinking was first through made-ground, then through loamy clay, peat, ballast, and London Clay, finishing in sandy clay. No special difficulties were met with, there being a freedom from water which would not have existed had it not been for the pumping that was being carried on in the cut-and-cover part of the work farther back. At the lower part of the sinking, especial care was necessary to keep the caisson upright, owing to the irregular nature of the ground, it being softer on one side. By proper attention to the weighting of the caisson, and by freeing the cutting-edge where the soil was stiffest, the required end was attained. The bottom of No. 4 shaft being 78 feet below high water, there was a good deal of water in the sandy clay. This, during the last few feet of the work, was not relieved by the pumping in the cut-and-cover, the work being at a lower level, and therefore a certain amount of pumping had to be done to keep the bottom part clear of water when putting in the concrete.

No. 4 shaft was sunk throughout in the open, but arrangements were made for putting in an air-tight floor (in case trouble should have arisen through water, and compressed-air, therefore, should have become necessary). The air-tight floor is shown in position in Fig. 3 (Plate XXVI.), but it was not used in sinking No. 4 shaft, though it was required in the other shafts. This air-tight floor consists of $\frac{5}{8}$ inch buckle-plates riveted to the top of 18 inches girders which are attached to the sides of the caisson. To give additional strength there are above ten girders 4 feet deep, which are at right-angles to the lower girders. Finally, these are crossed by two 12 feet girders. It will be noticed that the air-tight floor is placed higher than is usual in caissons used for sinking bridge-foundations and purposes of that nature. The reason of this will be obvious when it is remembered that provision had to be made for using compressed-air in constructing the tunnel itself, and in that case the floor would serve as a cover when the shield by which the tunnel was constructed was being driven through the shaft.

When the work of sinking the shaft was completed, it became necessary to shut off the water that came up from the sandy clay through the concrete. It will be understood that had not the pressure of this water been relieved it would have injured the concrete at the bottom of the shaft. The water was therefore allowed to escape through a pipe leading through the concrete and the water-tight floor to a sump, whence it was pumped to the surface. When the concrete had set and the work was

completed, the water was finally shut off by means of a valve which was placed in the pipe.

The No. 4 shaft occupied about nine months in sinking.

The Tunnel.—The work of tunnelling was commenced on the Kentish side (Fig. 2, Plate XXVI.). The exceptionally difficult nature of the work has already been alluded to; there being in one place but 5 feet 2 inches of gravel and sand between the top of the working and the bed of the river. In order to lessen the difficulty a considerable quantity of clay has been cast into the river from barges, so that it would settle above the gravel. In spite of this precaution, the difficulties which the engineers had to face were considerable, and it is due to the improved appliances of the present day that they have been overcome.

The tunnelling-shield, and the use of compressed-air in conjunction therewith, have enabled the engineers to proceed with this work in a manner that would have been formerly impossible. Fig. 4 (Plate XXVI.) is a longitudinal section showing the method of operating the shield, the men being shown at work. It is 19½ feet long over all, and is composed mainly of an outside shell consisting of four ⅝ inch steel plates. The interior space is divided, as shown, by two diaphragms consisting of vertical plates. These diaphragms are made air-tight, in order that an increased pressure of air may be maintained at the working-face over that which is required farther back. Up to the present time, however, this differential air-pressure has not been necessary. The air-locks are formed in the space between the two diaphragms, the doors being faced with indiarubber. The material excavated is carried through the diaphragms by the shoots, which are also provided with doors, and thus form air-locks. The part in the rear of the first diaphragm is generally described as the tail-of-the-shield; it consists of a built-up skin of plating. In front of the diaphragm, there is a second or inner skin: both skins being strongly attached to each other, thus forming a structure of considerable strength and stiffness. The two skins are brought together to form the cutting-edge, as shown in Fig. 4 (Plate XXVI.). The shield is divided horizontally by three platforms, thus affording four stages from which the working-face can be attacked; there are moreover three vertical partitions, which also act as stiffeners; and there are thus twelve compartments into which the front part of the shield is divided.

The shield was erected on the surface (No. 4 shaft was being sunk), and it was necessary, in order to save time, to proceed with the construction of

the shield simultaneously. The total weight of the shield is 230 tons. A dock was excavated in the earth close to the top of the shaft, and in this the shield was built. When the shield was erected, the two ends were closed by timber, so that the whole would float. The dock was next connected with the top of No. 4 shaft, a part of the caisson of the latter being temporarily removed for the purpose. Water was then admitted to the dock, of course filling No. 4 shaft at the same time, and the shield was floated out of the dock until it was above the shaft. By means of pumping, the water was removed from the shaft, and as the water-level sank, the shield gradually and safely descended until it assumed its proper position for commencing tunnelling operations, starting from the bottom of the shaft.

Some distance back from the cutting-edge (between 6 and 7 feet) there is a vertical screen depending from the top of each compartment. The space enclosed between this hanging-screen and the diaphragm forming the front of the air-lock forms a safety-chamber for the men to retreat to in case of a sudden irruption of water from the face, due to the air blowing-out. The water will not rise to the top of the enclosed space, being kept back by the imprisoned air, as in the case of a diving-bell. The men would thus be able to stand with their heads out of water until they could escape through the air-lock.

When working in gravel, sliding-shutters are used. These are pushed forward by means of screws to support the face after excavation, guides being provided for the purpose.

The shield is moved forward as the material is excavated by means of hydraulic rams placed at the rear (Fig. 4, Plate XXVI.). The completed segments of the cast-iron tunnel-lining form an abutment for the rams when pushing the shield forward. The head of the ram is bevelled, so that the pressure may come against the cylindrical part of the lining, and not bear on the flange, which might be broken. The rams are each 8 inches in diameter, and as the hydraulic pressure is 2 tons per square inch, a thrust of about 100 tons is obtained from each ram, so that the total pressure available for pushing the shield forward is about 2,800 tons. This has been exceeded at times, the hydraulic pressure having been raised to $2\frac{3}{4}$ tons per square inch. The rams are so arranged that they can be made to guide the shield, and correct any deviation that might result from inequalities in the nature of the soil or otherwise. This is effected by shutting-off any required number of rams on one side. Some trouble has arisen at times owing to the necessity of shutting-off some of the rams, and those remaining under pressure not being sufficiently powerful to impel the shield forward.

The cast-iron segments which form the tunnel-lining are erected inside the shield and fixed together, the tail-of-the-shield surrounding the last ring of lining put in place (Fig. 4, Plate XXVI.). The following ring of lining is then erected inside the tail, and the shield is pushed forward by the jacks as the work proceeds, and the erection of the rings is completed. For the purpose of placing the heavy iron castings in position, each segment weighing about 1 ton, two hydraulic erectors are used to lift the segments into place and hold them until they are securely bolted to the adjacent segments.

When tunnelling was first started from the bottom of No. 4 shaft, the face for about 2 feet deep at the top consisted of ballast which had been drained by pumping. There was about 1 foot of sand at the bottom, but the rest of the face was hard clay. Under these conditions, it was found possible to go some distance without using compressed-air. It was known, however, that the top of the tunnel would pass out of the ballast in a short distance, and it was therefore decided to drive a small top-heading in advance of the shield, timbering being used, rather than make use of the sliding-shutters before referred to.

The first permanent ring of the tunnel was erected on June 9th, 1893. The shield was then working down a gradient of 1 in 36 (Fig. 2, Plate XXVI.). By December 16th, 1893, a length of 477 feet of the tunnel had been completed, 191 rings having been erected. There was a heavy rush of water and sand into the heading that was being driven in advance of the tunnel, so that it soon became full of sand. The back of the shield was, therefore, timbered up, and the water rose to a height of 15 feet behind it. Further progress on the tunnel was then abandoned for a time, it being determined to carry on the remaining part of the tunnelling operations by the aid of compressed-air. A brickwork bulk-head was built across the tunnel, provided with two air-locks below, and a smaller one above, the latter being for use in cases of emergency. On March 14th, 1894, No. 3 caisson was sunk to its full depth, and on March 23rd compressed-air was put into the tunnel, after which trouble from water disappeared. An air-tight floor was built in the caisson of No. 3 shaft, similar to the floor described in connexion with No. 4 shaft. The tunnel-heading was now continued towards No. 3 shaft, and the shield finally reached that shaft early in May, 1894. The greatest pressure used during this portion of the work was 22 pounds per square inch.

Subaqueous Tunnelling.—Since September, 1894, the tunnelling from the south side has passed under the river. For the first 700 feet out of the

total of 1,221 feet of subaqueous tunnel (measuring from centre to centre of the two shafts on the river-banks), the progress was rapid and the work easy of execution: the soil being a fairly favourable loamy sand and clay. As much as 250 feet were completed in one month, the air-pressure ranging from 20 to 26 pounds per square inch. The tidal water of the river had very little noticeable effect when under the clay, but water came up from the land below, and compressed-air was used. After 700 feet had been constructed, ballast in direct communication with the river was worked into, and, naturally, the difficulties were at once immensely increased. The compressed-air began to escape very rapidly, so that the supply had to be greatly increased; the reserve compressors, which had been provided for that purpose, being all put to work. The artificial bed of clay which had been thrown down from hopper-barges covered the river-bed to a thickness of 10 feet, and extended for 75 feet on each side of the tunnel-line. This prevented the escape of air to a large extent, and gave protection against sudden runs of ballast into the shield. At this time there was but 5 feet 2 inches of ordinary river-ballast over the top of the shield.

At the present time (June 1st, 1895) about 950 feet out of the total length of 1,221 feet of river-tunnel has been completed.

The work is being carried out by the London County Council, under the control of Mr. A. R. Binnie, chief engineer to the Council. Messrs. David Hay and Maurice Fitzmaurice are resident engineers in charge of the work for the London County Council. The contractors are Messrs. Pearson & Sons; Mr. E. W. Moir is their engineer, and has designed most of the special plant used in carrying out the work; and Mr. E. H. Tabor is in charge of the machinery. Mr. G. H. Halden is responsible for the lines and levels in the tunnel.

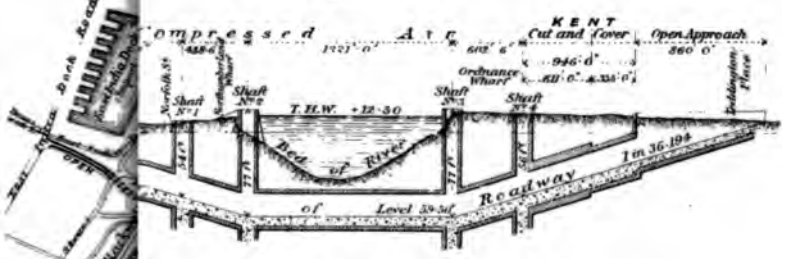
Mr. W. F. Howard proposed a vote of thanks to the Society of Arts for the use of their rooms for the meeting. Their best thanks were also due to the London County Council for giving them an opportunity of visiting the Blackwall tunnel, and also to their worthy President for presiding.

Mr. H. W. Hughes seconded the votes of thanks, which were approved.

A vote of thanks to the President for his services in the chair was then agreed to, and the meeting was closed.

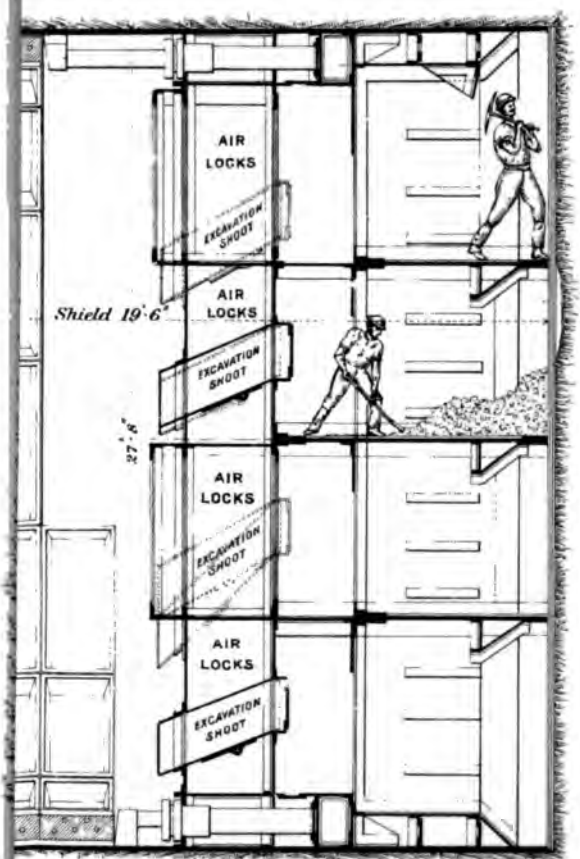
"kwall Tunnel"

FIG. 2.



GRAMMATIC SECTION OF TUNNEL

FIG. 4.



ION OF SHIELD

THE MINING INSTITUTE OF SCOTLAND.

GENERAL MEETING,

HELD IN THE HALL OF THE INSTITUTE, HAMILTON, JUNE 13TH, 1895.

MR. GEORGE A. MITCHELL, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and confirmed.

Mr. James Wallace, Wester Gartshore Colliery, Kirkintilloch, was elected a councillor in the room of Mr. Wm. Archibald, deceased.

The following gentlemen were elected:—

FEDERATED MEMBERS—

Mr. THOMAS BRUNTON, Grey Street, Shotts.
Mr. JAMES BURGESS, Dunaskin, Ayr.
Mr. JOHN M. CAIRNCROSS, 208, St. Vincent Street, Glasgow.
Mr. JAMES NISBET, 208, St. Vincent Street, Glasgow.
Mr. DAVID SMITH, Ferniegair Colliery, Hamilton.
Mr. JOHN THOMSON, Mainbank Colliery, Shotts.

DISCUSSION UPON MR. ALLAN STIRLING'S PAPER ON "WATER-TUBE BOILERS."*

Mr. THOS. WILLIAMSON (Wishaw) wrote that a Stirling boiler at the Glasgow Iron and Steel Company's works had, after three months' running, given complete satisfaction in all respects.

Messrs. JOHN LAWSON & SONS (Kilmarnock) wrote that the Stirling boiler referred to in Mr. Stirling's paper, which had replaced a Cornish boiler, was saving from 20 to 25 per cent. of fuel, that the fuel used was a lower quality, costing 20 per cent. less, and that after two months' use they were thoroughly satisfied with the boiler.

* *Trans. Fed. Inst.*, vol. ix., page 2.

Mr. JARDINE (Motherwell) wrote that the Stirling boiler erected at the Dalzell Steel-works had been working over three months with a cold feed, and the water of very bad quality, depositing sediment in the form of mud in the mud-drum. The boiler had to be blown down half-glass once in twenty-four hours. The mud-drums had been examined twice since the boiler was erected. The front drum had a thin coating like enamel all round, caused by the evaporation. The centre and back drums and tubes were clear of any deposit. He (Mr. Jardine) had no hesitation in saying that, if the boiler worked as efficiently as it had been doing, it was one of the best boilers at the Dalzell works.

Mr. FRED. J. ROWAN (Glasgow) wrote that the title of Mr. Stirling's paper was misleading, because the paper was not written upon the subject of "Water-tube Boilers" in general, but was merely on that of the "Stirling" water-tube boiler in particular. The general subject was one of very great interest from an engineering point of view, and one of very much greater antiquity and development, in this country at any rate, than Mr. Stirling's short references to history would lead anyone to suppose. It might, nevertheless, surprise some members to hear that perhaps the earliest attempt to make a water-tube boiler, of which there was any record, was that of Mr. Woolf (the reputed inventor of the compound engine) in 1803. Following him came Messrs. Gurney and Hancock in 1827 and 1831 respectively, both of these inventors having designed and introduced such boilers to meet the requirements of traction by means of steam-carriages on common roads. Mr. Jacob Perkins, the inventor of the steam-gun, also patented a boiler in 1831, and a few years afterwards Mr. Thomas Craddock and the famous Earl of Dundonald invented water-tube boilers. Since that time an almost endless variety of designs had been brought forward, although for the most part inventors had preferred some modification of the arrangement which showed tubes slightly inclined from the horizontal position and connected to headers at each end, such modifications being fairly illustrated in the Root, Belleville, and Babcock & Wilcox boilers of to-day. It was quite true that in spite of all these inventions the water-tube boiler never became popular, and in fact never had a chance of anything like general introduction until quite recently; but this only showed how slow engineers were to appreciate what was really good. His (Mr. Rowan's) interest in the subject dated from about 1860, when he was engaged in the construction of the sectional boilers and compound engines invented by his father in 1858 and 1860. He (Mr. Rowan) read a paper on "The Design and Use of Boilers" before

the British Association for the Advancement of Science in 1878.* This paper contained a full account of the principles of the subject. The *Transactions* of the Institution of Engineers and Shipbuilders in Scotland † contained a paper which he had written on "The Introduction of the Compound Engine, and the Economical Advantage of High Pressure Steam," etc. This paper contained a summary of the examples of his father's water-tube boilers which had been introduced between the years 1858 and 1874. So that at the time when Mr. Stirling was not yet convinced as to the merits of the water-tube system, he (Mr. Rowan) was publishing the results of sixteen years' experience with one class of examples of that system. The first proposal to construct a boiler with vertical water-tubes of small diameter, slightly bent at their top and bottom ends so as to branch radially from cylindrical chambers or drums at each end, being expanded directly into the tube-plates which formed part of the diameter of the drums, was contained in his (Mr. Rowan's) father's patent of 1869. One arrangement of this boiler was practically the Stirling boiler placed upside down, two grates being disposed between the three water-drums which were below the grate level, whilst the upper drum was used for collecting steam and separating it from priming water. The advantages of the Stirling boiler, which Mr. Stirling ranged under the Nos. 1 to 12,‡ were (with the exception of Nos. 6 and 12) equally applicable to many other water-tube boilers, and in fact some, if not most, of them had been originated in the designs of other boilers, as for instance the boilers of 1869 to which he had referred. He (Mr. Rowan) agreed with Mr. Stirling in strenuous belief in the merits of the water-tube system when it was properly carried out—in fact he believed that it was the rational system of construction for steam boilers, but he did not think that the Stirling boiler, whatever its advantages might be, could be considered as by any means the last word on the subject. He did not know if he was correct in supposing that this boiler was the invention of Mr. Stirling, because he recently noticed in the *Engineer*,§ amongst the records of American patents, an exactly similar design which had been patented on December 22nd, 1893, in the name of Mr. H. S. Pell, of Akron, Ohio, so that it appeared as if some one else were laying claim to this design. With regard to the special features of Mr. Stirling's boiler, he could not accept without question the statement that it "occupies less space than any other boiler." He supposed that meant for a given power. This

* *Report*, 1878, page 712. † Vol. xxiii., page 51.

‡ *Trans. Fed. Inst.*, vol. ix., pages 5 and 6.

§ 1894, vol. lxxviii., page 138.

was a matter which could be settled only by a comparison of its dimensions with those of other boilers, and the paper did not afford the materials necessary for such a comparison. Mr. Stirling also said that the "boiler is supported on wrought-iron columns and is sustained independently of the brickwork," but the illustrations give no hint, in either view, of such an arrangement, and their incompleteness prevents the members from forming any opinion as to that arrangement. On two other points he was inclined to question Mr. Stirling's statements, viz., as regards the circulation of the water between the front and middle drums, and as regards the removal and renewal of vertical tubes. Mr. Stirling said that "the circulation of water between the front and middle drums passes freely through the numerous tubes which connect them below the water-level,"* but such tubes are not clearly shown in Fig. 1. If the inclined tubes rising from the water-drum were relied upon, the action would be extremely uncertain, as steam and water must be constantly rushing upwards through these tubes, and it was plain that water could not be expected to pass downwards against that current. Short tubes of small diameter passing directly across from one drum to the other seemed to be indicated, but in this case the middle drum must be pierced on a part of its circumference which is beyond the tube-plate, and this must detract from its strength. The only alternative to that would be to extend the tube-plate, which is thicker, and of course heavier, than the rest of the drum, to an inconvenient size. The same remark would apply if a number of small tubes are used to connect the steam spaces of the several drums. The renewal of a tube in one of the inner rows of upright tubes was not an easy matter unless the tubes in the outer rows were placed at some distance apart from centre to centre. More frequently than not, the removal and renewal of a tube in one of the inner rows of such a boiler involved the cutting out of at least one tube in the outer row. This was a matter of experience in the handling of such boilers, and no doubt Mr. Stirling could state how many tubes he had replaced, and what the mode of procedure had actually been in such cases.

Mr. J. S. DIXON (Glasgow) said that one of the most valuable features of the Stirling boiler-tubes, from the back drum downwards, was that they formed a heater for heating the water before there was a temperature to produce steam, and the deposit was left in the mud-drum. In his experience with Babcock & Wilcox boilers, the great *sine qua non* was clean water, and, having that, he would have no hesitation in advising anyone to put them in.

* *Trans. Fed. Inst.*, vol ix., page 5.

The PRESIDENT said that it was an unsatisfactory feature of the paper that no tests were recorded of a Stirling water-tube boiler, showing the fuel consumed for the steam produced. Perhaps Mr. Stirling would be able to give them some tests from boilers now in use.

The discussion was then adjourned.

DISCUSSION UPON MR. W. C. MOUNTAIN'S PAPER ON "THE DESIGN, EFFICIENCY, AND APPLICATION OF ELECTRIC-MOTORS FOR TRANSMISSION OF POWER IN MINES."*

Mr. WALTER DIXON (Glasgow) regretted that the writer fell into the track of several other writers on similar subjects, by what are necessarily inadequate and too brief explanations of the ways and habits of electricity and its behaviour in a dynamo or motor. He thought that the value of electricity as a motive power was now fully recognised, and that mining engineers were not so much interested in how the machines were made as in their application. It was satisfactory to know that there were in Scotland several plants of 100 horse-power, and one or two larger; while across the Border, plants of 500 horse-power were being employed. With electrical machinery as at present designed, there was not much to be desired in the way of greater efficiency, and one was therefore justified in adopting its use, without waiting, as we are apt to do, for the invention of improved methods. Both dynamos and motors are being used which, while retaining all the good points of the shunt, series and compound-wound machines, overcome the troubles in working which had hitherto been associated with these machines. In other countries large central-station plants of from 1,000 to 10,000 horse-power were now being used or erected for the distribution of motive-power over large areas, and in districts like our own, where pits and works are in such close proximity, it is probable that at some future date it may be considered practical and economical to concentrate at central points the power obtained from high efficiency engines, and to distribute the same to users over large areas, in the same manner as hydraulic power and gas are distributed at the present day over more limited areas.

A vote of thanks was accorded to Mr. W. C. Mountain for his paper, and the discussion was closed.

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* *Trans. Fed. Inst.*, vol. ix., page 14.

DISCUSSION ON MR. A. M. GRANT'S "REMARKS ON WINDING-ENGINES."*

Mr. GRANT said that the most powerful example of a modern high-speed horizontal winding-engine in Scotland was erected about two years ago at the Newbattle collieries of the Lothian Coal Company. The cylinders are 40 inches in diameter by 7 feet stroke, working with a steam-pressure of 100 pounds per square inch. The cylinders are steam-jacketed, and supplied with steam direct from the boilers. The pistons are of the hollow-box form, fitted with Mather & Platt piston-rings. The piston-rod is carried through both cylinder-covers, and is supported at the back by a single slipper guide-bar. The connecting-rods, cranks, and crank-shaft are of best hammered scrap-iron, and the crank-pin is of forged steel. The crank-shaft is 18 inches in diameter, with bearings 16 inches in diameter and 25 inches long. The drums (21 feet in diameter and 8½ feet wide) are made of cast-iron, with steel-plate lagging, ½ inch thick, and fitted with brake-rings on each side. The four upright brakes are arranged to work either by steam or foot power. The cylinders are fitted with Cornish double-beat valves and gear of the usual Kilmarnock type, the steam-valves being 11 inches, and the exhaust-valves 14 inches in diameter respectively. The valve-gear is fitted with an automatic cut-off, controlled by a high-speed governor worked from the crank-shaft. The shaft *A* (Fig. 10, Plate XXVII.) is driven by bevel-gearing on the drum-shaft, and drives the high-speed governor *B*. The movable sleeve, combined with the governor, is connected by levers and rods *D*, *D*, *D*, to the cross-shaft *E*, and from the shaft *E* by means of levers to the upright rods *F* placed between the steam valve-chests on each cylinder. The upper ends of the rods *F* work through a guide *G*, and the projecting points at the top of the rods are connected by the rods *H*, *H* to the ends of the trip-cams or levers of the steam-valves *K*, *K*. When the speed of the engine and governor increases, the vertical rods *F* are drawn downward, the trip-levers *K*, *K*, gradually drawn out of contact with the steam-valve *L*, and the trip action taking place sooner or later, as the engine is going faster or slower. The shaft *N* and levers *M* are connected by means of levers, rods, and handles to a point convenient for the engineman, so that by means of a lever handle he has the cut-off always under control, and can vary it as desired when the engine is running, whether it be acted upon by the governor or not.

* *Trans. Fed. Inst.*, vol. viii., page 390; vol. ix., page 141.

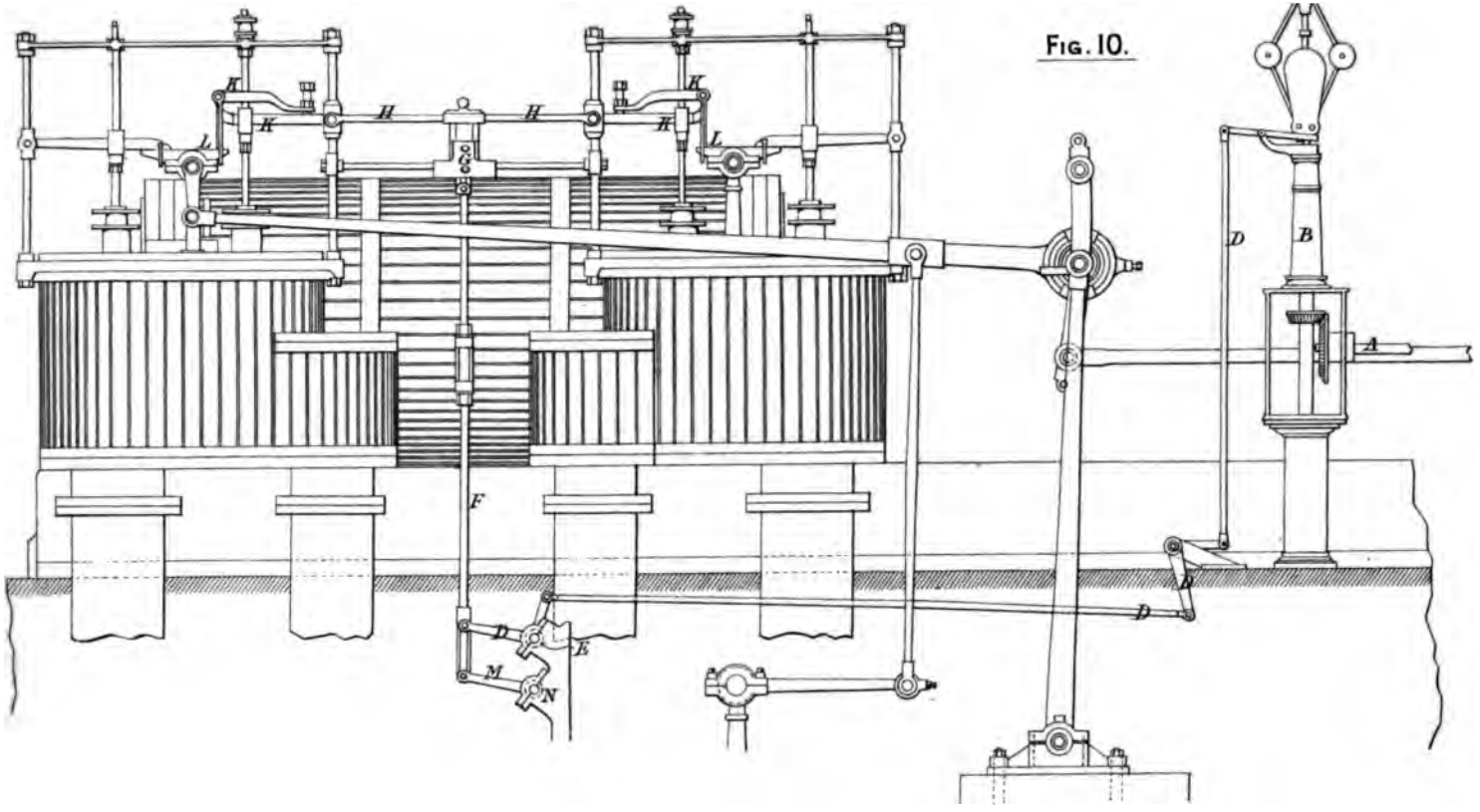

FIG. 10.

Mr. JAMES FREW (Dunaskin) asked Mr. Grant what was the diameter of the relief-valves applied to the cylinders of the Newbattle winding-engine; whether he had tried the capability of release with the engine running at the maximum speed and the automatic cut-off gear in action, with the steam instantly shut off at the throttle-valve; and also whether the release was ample if full steam were instantly applied to stop the running of the engine in cases of emergency? He also asked whether the attendant had a separate control of the automatic cut-off gear while the engine was in motion, or whether the gear was controlled by the governor alone, subject to adjustment? Could Mr. Grant give comparative results of the steam used with and without the cut-off gear in action, with full loads? Did the speed of the engine decrease when the cut-off gear was in action, and if so, how much, and what was the time of running, depth of shaft, and weight of load lifted? Did Mr. Grant consider that a winding-engine, fitted with double-beat valves and tappets, as durable as one fitted with slide-valves and the usual link-gear; did the wear of the tappets not increase the back pressure on the engine much more readily than the wear of the gear on slide-valve engines? Would Mr. Grant state roughly the difference of cost of a winding-engine with two cylinders 30 inches in diameter fitted with good balanced slide-valves, as compared with Cornish double-beat valves? He (Mr. Frew) considered that there was no difficulty in designing and balancing slide-valves for cylinders up to 36 inches in diameter for a steam-pressure of 80 pounds, so as to be under the control of the attendant when reversing.

Mr. GRANT said that the relief-valves were 3 inches in diameter, and he had tested them in every way. They had relieved the pressure at 300 lbs., and they had found them under every possible circumstances to be perfectly ample for their intended purpose. The automatic cut-off gear could be altered in any way desired, and could even work against the governor. The speed of winding was decreased when the automatic cut-off gear came into action. The depth of the shaft was about 1,710 feet. The weight of coal in each cage was 10 tons, and the total weight lifted from the bottom at the beginning of the winding was about 18 tons, and the time occupied by a single wind was about 48 seconds. The boiler-pressure was 100 lbs. per square inch. He had no doubt that Cornish valves would wear more quickly than common slide-valves, but it was an easy matter to keep them in repair. With cylinders 30 inches in diameter, a Cornish valve was superior to any balanced slide-valve. The difference in cost would vary from £150 to £180.

Mr. D. BAIRD (Leven) agreed with Mr. Grant that if the diameter of the cylinder exceeded 26 inches the use of Cornish valves became almost a necessity. Would Mr. Grant state whether or not there were only two classes of valves suitable for winding-engines, viz., slide and Cornish valves? The heavy wear-and-tear of Cornish valves had been mentioned as one of their weak points.

Mr. GRANT said that he had never applied piston-valves to winding-engines. They were often used in rolling-mill engines, but there was great objection to them, as they were difficult to keep in repair.

Mr. BAIRD said it would not be difficult to renew piston-valves if brass liners were used.

Mr. GRANT could not see the slightest difficulty in applying piston-valves to a winding-engine.

Mr. GLEN (Airdrie), said that there had been piston-valves working at Rosehall colliery for about fifty years. They were easily handled.

Mr. GRANT said that where Cornish or piston-valves were used an escape-valve should also be provided as a necessary precaution.

Mr. BAIRD said that there could be no objection to a safety-valve being used. It would not in any way affect the efficiency of the engine.

The discussion was adjourned.

DISCUSSION OF MR. J. G. BARCLAY'S PAPERS ON "THE BARCLAY CROWN-VALVE EXHAUST-GEAR"* AND ON "AN OVERHANGING BEAM PUMPING-ENGINE."†

Mr. JOHN GLEN (Airdrie) said Mr. Barclay claimed that, by raising the air-pump to the floor-level and attaching it farther back on the beam, he thereby shortened the stroke by one half. With the 6 feet stroke Mr. Barclay said that there was 3 lbs. pressure due to that head of water on the air-pump bucket. Now, if there were 6 feet of water on the bucket when it began to rise, that head of water had to be discharged during the whole length of the stroke against the atmosphere; that is, 3 + 12½ lbs. per square inch of vacuum under the air-pump bucket would give 15½ lbs. to be overcome at the beginning of the stroke, but the 3 lbs. would gradually diminish as the upstroke proceeded. Mr. Frew said Mr. Barclay did not mean that 6 feet of water was on the top of the bucket, but that there was 3 feet of water on the bucket

* *Trans. Fed. Inst.*, vol. viii., page 396; vol. ix., page 142.

† *Ibid.*, vol. viii., page 397; vol. ix., page 142.

and 3 feet under it. This statement is reasonable and in keeping with Mr. Frew's theory that the 2½ lbs. lost in the condenser makes the water follow the bucket. He (Mr. Glen) would like to know how Mr. Frew will do work with this 2½ lbs. which he says is lost. The back pressure in the condenser is due (all joints being air-tight) to the temperature of the water in the condenser, and varies accordingly. By the permission of the Coltness Iron Co., and the assistance of their engineer (Mr. McGregor), some experiments were made with the air-pump and condenser (Fig. 4). A water-gauge *A*, 50 inches in

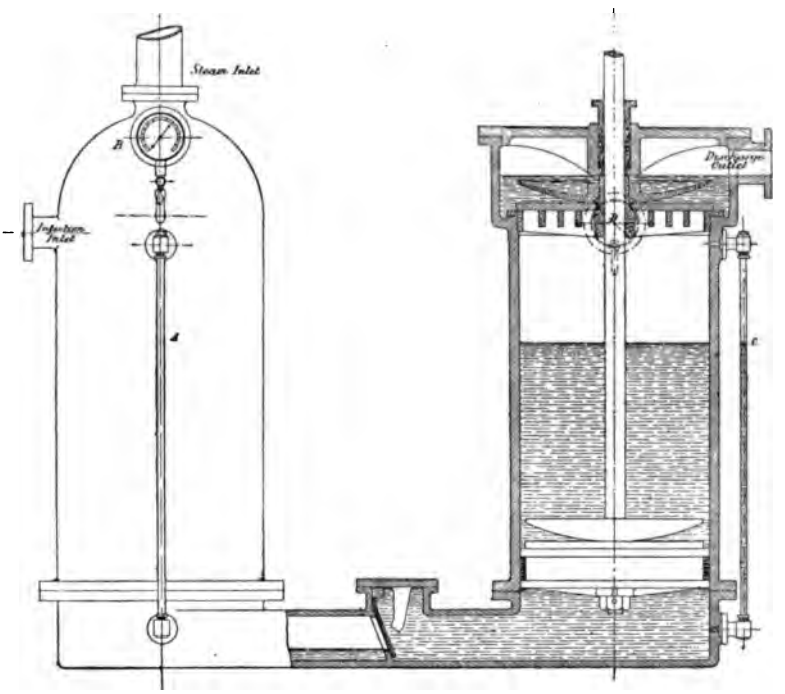

Fig. 4.

length, was placed on the condenser, and a tested Bourdon vacuum-gauge *B* at the top of the water-gauge, while a similar water-gauge *C* was placed on the air-pump with a tested combined vacuum and pressure-gauge *D*. During the experiment, it was found that the lowest gauge-cock (fixed as shown in Fig. 4) had been placed too high to record any readings, as the water did not appear in the gauge-glass until the stroke was about completed, showing clearly that the whole of the water was displaced by the force of the exhaust steam in the condenser, and did not reappear in the glass until the next stroke was nearly completed.

It will be seen from the position of the air-pump bucket (Fig. 4) that the whole of the water that fell in the condenser during the complete (or double) stroke of the engine was forced into the air-pump. Before explaining what took place in the upstroke, it would be well to put Mr. Frew's theory to the test. Supposing that the water was at the same level in the condenser as it was in the air-pump, and that the bucket begins to rise, the water would fall in the condenser to the level of the bottom of the bucket (due to gravitation) and remain there, because that is the point of separation, where the 2½ lbs. is given out as well as in the condenser. As water boils at a low temperature under a vacuum, the greatest amount of evaporation takes place there, and prevents the 15 lbs. being got, which is the average weight of the atmosphere, and without which Mr. Frew would not get the water to follow the bucket. In the up-stroke of the bucket, the space between the surface of the water, which is 16 inches, and the discharge-valve (if this vapour was a permanent gas such as air) would act as a rarefied air-compressor with this 2½ lbs., but no such result takes place. By shutting the bottom cock of the gauge-glass and leaving the combined vacuum and pressure-gauge *D* attached to this space, the gauge gradually fell back to zero, and only showed ½ lb. of pressure during the discharge. At this point of the discharge the engine exhaust takes place, and the vacuum gauge on the condenser fell back from 25 to 21 inches, at the same time as the water was raised 12 inches in the air-pump. When the bucket returns from the top it leaves a vacuum of 12½ lbs. between the discharge and the bucket (the warm water lying on the top of the bucket intervening again); at the bottom the exhaust takes place again, and the gauge falls back this time from 25 to 19 inches, the water is raised this time to 28 inches, the difference in the quantity being due to the bulk of the bucket and the water on the top of it, and the difference in the fall of the gauge is due to the greater resistance encountered in forcing the water through the bucket, as shown in Fig. 4. It may be mentioned that this engine pumps direct from the cross-head, the cylinder being 3 feet in diameter and 6 feet stroke. The air-pump is 25 inches in diameter and 3 feet stroke. It works about four strokes per minute with a mean steam-pressure of 13 lbs. per square inch.

Mr. D. BAIRD (Leven) said that Mr. Glen had given the members very interesting details as to the position of the water in the air-pump and condenser, but he did not think that it affected the question before the members. The difference between an air-pump with half a stroke and one with full stroke would in practice be very little, and hitherto he had

paid no attention to that factor. Mr. Barclay's design of a pumping-engine had, in his opinion, only one good point to recommend it to mine-owners, viz., that by such an arrangement the cylinder was vertical, and consequently was in the best position for the reduction of wear in the piston and cylinder-walls. But, even with this advantage, it was not, in his opinion, an efficient engine for underground use. The advantage of a vertical cylinder was after all not so very great, and they had the opinion of many engineers that, with well-designed pistons, the wear-and-tear on a horizontal cylinder was very small. The addition of the flywheel to the Barclay surface-type of pumping-engine was, he thought, a mistake, as it prevented the necessary pauses at the termination of each stroke. This engine, situated as it was underground, had the very important defect of all underground steam-pumps. A large portion of the steam was condensed on the way to the engine, and to this was added the loss owing to the moist condition of the steam when it arrived at the engine. No matter how well trapped the steam-pipes might be, there was sure to be a large percentage of moisture in the steam, which was a sure cause of lessened efficiency. To what extent this could be remedied by super-heating the steam had not, he thought, been proved by experience, but it was a matter which should be tested. Some engineers who were manufacturing steam super-heaters at present claimed 20 per cent. of gain on this point for surface engines, but had not had sufficient experience to name the per-centage for underground engines. The pump shown in Plate XXIII.* was of the packed bucket type, which for 1,200 feet of a vertical throw was not suitable. Double rams, no doubt, could be adopted with side rods, which would give better efficiency. The engine was far too near the shaft and would be kept clean with difficulty.

Mr. J. S. DIXON (Glasgow) said that the Barclay crown-valve exhaust-gear was fitted to the winding-engines at Hallside and Hamilton Palace collieries, and they worked excellently. The engine-diagram had been improved, partly because the improved trip-gear opened and shut the valves more quickly, and partly owing to the cut-off being better adjusted. There was no doubt that this gearing was an improvement on the Cornish valve-gear. At Hallside, where the shaft was 1,380 feet deep, the engineman could wind considerably faster than he could prior to the application of the Barclay valve-gear.

Mr. GRANT asked Mr. Dixon whether he could state the amount of back pressure?

* *Trans. Fed. Inst.*, vol. viii.

Mr. DIXON replied that, in both cases, the back pressure was very considerable, and had been reduced by the use of the Barclay valve-gear.

Mr. GRANT stated that some indicator-diagrams taken at the Hamilton Palace colliery showed no back pressure. He thought that the valve-gear had got out of order, and suggested, if the valve-gear had been adjusted, that similar results would have been obtained.

Mr. DIXON said that the improvement was partly due to both causes, but the Barclay gear was an improvement on the old system.

Mr. GRANT considered that the Barclay gearing was absolutely unnecessary. A similar result was obtained at Hamilton colliery by keeping the exhaust-lever close to the cam.

The discussion was adjourned.

APPENDICES.

I.—NOTES OF PAPERS ON THE WORKING OF MINES, METALLURGY, ETC., FROM THE TRANSACTIONS OF COLONIAL AND FOREIGN SOCIETIES AND COLONIAL AND FOREIGN PUBLICATIONS.

A FURNACE WITH AUTOMATIC STOKER, TRAVELLING GRATE, AND VARIABLE BLAST, INTENDED ESPECIALLY FOR BURNING SMALL ANTHRACITE COALS.

By ECKLEY B. COXE. *Transactions of the American Institute of Mining Engineers, 1893, vol. xxii., pages* 581-606.

The difficulties of burning small sizes of anthracite coal are such that large quantities of such fuel have in the past been thrown away. On the formation of a Commission by the Legislature of Pennsylvania for the purpose of investigating the "Waste of Coal Mining," attention was directed to the most economical method of burning small sizes (from $\frac{0}{16}$ to $\frac{3}{16}$ inch diameter) of anthracite coal, and as an outcome of experiments carried out at Drifton, the furnace and stoker herewith described was designed by Mr. Coxe. In a general way it may be said that the proper conditions required to be fulfilled were: (*a*) To ignite the coal, and burn it up without mixing it with fresh fuel, that is, that fresh fuel would not be added to the already partially consumed coal. (*b*) To have the furnace so arranged that the combustion would be continuous and uniform.

The furnace consists essentially of a travelling grate, moving at the rate of from 3½ to 5 feet per hour to the back of the furnace. The coal is brought to a feed-hopper, and passes down by gravity over a fire-brick, called the "ignition brick," and under a regulating-gate, on to the travelling-grate. When the furnace is heated, the ignition-brick becomes hot, and the coal passing over it reaches an incandescent state, often becoming hot enough to ignite before passing on to the grate. Under the grate are a number of air-chambers, made of sheet-iron, which are closed on all sides except the top. Blast from a fan is blown into the second air-chamber from the right. The air-chambers are open on the top, but the partitions between are covered by plates of such a width, that no matter what may be the position of the grate-bars, there is always one resting on the plate. Consequently, air cannot pass from one chamber to the other except by leakage along the bar. The consequence is that the air-pressure in each successive chamber is smaller than in the adjoining one, passing from the front to the back of the furnace.

As the thoroughly ignited coal passes slowly over the second compartment (where the air-pressure is at a maximum) it burns briskly; then slowly passing over the third compartment, where the air-pressure is less, and better suited to the combustion of the thinner layer of partially consumed coal, the bed continues to diminish in carbon, and to be subjected to less blast, until finally the hot ashes are cooled off by a very gentle current of air. H. W. H.

COAL-DUST QUESTION IN ENGLAND.

Note sur la question des Poussières en Angleterre. By L. AGUILLON. *Annales des Mines*, 1895, *series* 9, *vol. vii.*, *pages* 535-559.

The author confines himself to the consideration of the results of the Royal Commission on Explosions from Coal-dust.

The commission undertook the consideration of recent large accidents which had been by some attributed to coal-dust. They examined on the spot four explosions which occurred while they were sitting, and directed new experiments on coal-dust to be made by Mr. H. Hall.

The conclusions of the commission are said not to be such as to impress opponents of the coal-dust theory. The only new elements that they had to consider were the experiments of Mr. Hall, which the author thinks differed little, so far as concerns the rational and definitive solution of the question, from the experiments made by the Prussian Commission at Neunkirchen.

The conclusions of the commission are said to resemble the decision of an arbitrator who decides on the evidence of opposing witnesses. Scientific questions cannot be decided in this way. The author thinks that the divergence of opinion between the supporters and opponents of the coal-dust theory may be more apparent than real, because they do not appreciate the difference between inflammation and explosion.

The observations and conclusions of the commission are then set forth and commented on. With reference to the distance traversed by some explosions, an account by the author of the explosion at Seaham in 1880 is referred to, and it is stated that the flame in that explosion only exercised action within a very restricted area. [This is an incorrect statement, probably based on imperfect information.—EDITOR.]

With reference to the lack of indications of violence for some distance on each side of shots to which certain explosions were attributed, it is remarked that the commission appear to have forgotten the experiments of Messrs. Mallard and Le Chatelier on gas explosions in tubes and the difference of the effects according to the point of inflammation in the tube. [The commissioners do not state the most important conclusion drawn from the spaces free from violence with great damage on each side; such conclusion did not relate to the question of gas *versus* coal-dust, but was simply that they indicated the place where the explosions commenced, and there is nothing in the experiments of Messrs. Mallard and Le Chatelier opposed to this conclusion.—EDITOR.]

The appendices contain accounts of the accidents at Brandon and Camerton collieries and of Mr. Hall's experiments. It is doubted whether these experiments produced explosions as opposed to inflammations, and they are held not to authorize the deduction that the inflammation or explosion would continue indefinitely.

W. N. A.

COPPER-MINING IN TENNESSEE, U.S.A.

Ducktown, Tennessee, Copper-mining District. By WM. M. BREWER. *Engineering and Mining Journal* (*New York*), 1895, *vol. lix.*, *page* 271, *with one figure.*

The Ducktown district is situated in the extreme south-eastern corner of Tennessee, and within a very short distance of the meeting point of the three states of Tennessee, Georgia, and North Carolina. The district is bounded on the east by the Blue Ridge mountains, on the west by the Smoky range, with Frog mountain near the south-western extremity. The Ducktown Copper, Sulphur

and Iron Company four years ago purchased the copper-mining property originally owned by the Union Consolidated Mining Company. The last-named company, several years ago, mined out a body of black copper ore, which occurred underlying the gossan, and it is the purpose of the present company to commence underground explorations in order to ascertain the extent of the body of unaltered pyrites, which is expected to occur below the gossan. The new company is prospecting on the Callaway estate, and extracting ore from the Mary mine, which furnishes the supply for the furnaces.

On the 20-fathoms level in the Mary mine, from which this ore is being mined, the workings show in many places the body of unaltered pyrites to be 90 feet in thickness. This ore carries an average of about 3½ per cent. of copper and 27 per cent. of sulphur, with no arsenic or other injurious element. Consequently the matte from the first smelting is very clean, and averages 46 per cent. of copper, a most unusual proportion to be contained in the matte after the first smelting from an ore of so low a grade. During the four years that the present company has been in operation it has done much experimental work, under the superintendence of Mr. T. Carter, formerly of Shropshire, England.

Sixty shelf pyrites-burners for roasting ore were erected, with a capacity of 3,000 pounds each per day, and arranged with six shelves in each furnace, 6 feet by 10 feet. Each charge of fine pyrites now requires 48 hours. J. W.

THE DOLGOROUKY COPPER MINES, RUSSIA.

ANON. *Indian Engineering*, 1895, *vol. xvii.*, *pages* 44-45.

The Dolgorouky mines are situated on the Steppe in the districts of Boulinsk and Bourlinsk, some 65 miles south of the town of Orenburg. The mineral rights belong to the Russian Government, which has the power to grant concessions for mining; otherwise the great mineral wealth of the country would remain undeveloped, the native Khirghiz being a very superstitious race.

The rocks are chiefly of Permian age, well stratified. The ores are oxidized compounds of copper, the green and blue carbonates, usually termed malachite and azurite, with small agglomerations of copper-glance in places.

At Kisul-Bei mine the average yield of copper is 10·1 per cent., the bed having a thickness of from 1½ feet to 2½ feet. At Kouchouk-Bei, the bed is about the same thickness, the yield being only 4·5 per cent. of copper. At the Cir-Koul mine, ore varying from 10·5 to 21·5 per cent. of copper has been removed.

Complete analyses of the ore show :—

	Cir-Koul.	Kisil Adeur.
Copper oxide	13·13 ...	10·41
Ferric oxide	10·72 ...	18·71
Carbon dioxide	1·36 ...	3·27
Water of combination	3·89 ...	1·50
Silica	71·00 ...	66·07
Totals	100·10 ...	99·96

The ores from all the mines are similar as regards their freedom from injurious elements, only a trace of sulphur being present and arsenic being absent. No expensive roasting is required; in a single fusion metallic copper can be obtained. The cost of mining and transport to furnace is about 15s. per ton, and the cost of treatment is about the same. J. W.

ELECTRIC HAULAGE AND SIGNALLING.

The Engineering and Mining Journal (New York), 1895, *vol. lix.*, *page* 33, *with two illustrations.*

The Western electric locomotive is waterproof, has two slow-speed motors connected in parallel, each motor operating an axle by a single reduction with cut gears. Armatures are ring-wound, thus permitting the removal of a coil without tearing the whole armature to pieces. In the machine illustrated, the weight will guarantee a haulage of 75 tons without slipping its wheels on a straight level track. The dimensions of the locomotive are:—Length, 8½ feet; width over all, 54½ inches; height above rail, 45 inches; weight, 15,600 lbs.; width of gauge, 36 inches.

The Western signal bell is a positive stroke gong, simply constructed, has no winding mechanism, and is non-vibrating. No mistakes can be made by the engineer, as only one blow can be struck for each complete turn of the handle, and the handle cannot be moved backward to ring the gong. One bell will suffice for several levels, but a separate switch is required on each level. J. W.

A NEW ELECTRIC MINE HOIST.

ANON. *The Engineering and Mining Journal (New York)*, 1895, *vol. lix.*, *page* 201, *with illustration.*

An electric hoist has been erected at the Priceburg mine, at Dickson, Lackawanna county, Pennsylvania. It forms part of a complete electric plant being placed at the mine, including an 80 horse-power electric locomotive and a 100 kilowatts generator. The hoist has two motors, the advantages of which arrangement are fully explained; it is a hoist of the friction type, having internal friction-gear. It is to be used from the upper end of the slope, for hauling cars out of the butt entries into the main entry and lowering them down to the level track, where they will be made up in trips for the locomotive to haul out to the shaft. The advantage of this arrangement becomes more apparent when it is considered that the hoist is to be placed 5,000 feet from the power-house upon the side of a mountain, where it would be practically impossible to use a steam plant, as there is no water to be had conveniently, and it would be difficult and expensive to carry coal for firing the boilers. J. W.

THE GENESIS OF ORE-DEPOSITS.

By F. POSEPNY. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxiii.*, *pages* 197-368 *and page* 587.

The author of this exhaustive paper begins by briefly summarizing the various systems of classification employed up to the present time, and comes to the conclusion that a stable and complete system can only be framed when all the controlling facts—in other words, all the ore-deposits—are accurately known. This is not likely ever to be the case. New observations are being constantly made in mining, and these often obliterate the old ones, so that they cannot be verified and compared. It is, however, absolutely necessary, in a field so complicated as that of ore-deposits, to have some general system, comprising all known results.

In framing such a system, the characters of form being the most obvious and the most familiar to the miner would be naturally emphasized, while genetic characters would be left in the background. But this ought not to check genetic investigation, or the advancing recognition of real relations.

A genetic system must involve hypotheses, and may not, for a while, be practically useful; but in time it will, like every other cultivated branch of geology, assume more permanent forms.

The principal genetic distinction is doubtless between deposits contemporaneous with the country-rock and those subsequently formed in it.

There are two main groups of mineral aggregates--(1) rocks; and (2) mineral deposits.

The minerals of the first group belong to it as native and original, and may be designated as idiogenous. Those of the second group are foreign to the rocks in which they occur, and may be designated as xenogenous.

If we find in the midst of stratified formations ores lying exactly between two strata it is not conclusive proof of their sedimentary or precipitative origin. This must be proved in every given case, for in the present state of knowledge we cannot understand how the metallic sulphides, so characteristic of ore-deposits, could be formed in that way.

As to the eruptive rocks, we do not know what they once were, for we study them only from the moment of cooling. But we observe at once that iron occurs primitively in these rocks in the form of magnetite. This idiogenite of the eruptive rocks can be detected without chemical aid; but with such aid we find traces of other metals, and this leads us to surmise that the eruptives have brought a whole series of heavy metals from the barysphere into the lithosphere, and it looks as if the metals of the ore-deposits originally belonged to the barysphere.

With regard to the mineral deposits or xenogenites, it is apparent that they must either have found a space ready to receive them or have made room by driving out an original mineral. We know that small pores or cavities exist in substances of the greatest density, since we can, by adequate pressure, force quicksilver through them. Moreover, we find larger cavities or blow-holes in the eruptive rocks.

Mineral deposits formed upon the walls of cavities by liquids circulating within them usually have a characteristic structure, for which the term "crustification" is proposed. These crusts generally occur concentrically in regular succession, covering not only the cavity-walls, but the surface of every foreign body in the cavity. Sometimes the crusts have undergone a secondary alteration (carbonates are replaced with silica, etc.). The crustification is thus made less distinct, or is even obliterated. As a general rule, however, crustification is a characteristic feature of cavity-filling.

The cavities are formed either by mechanical forces, called spaces of dislocation or discission, or by chemical forces called spaces of corrosion or dissolution, and these two must be sharply distinguished.

Spaces of dissolution naturally occur in soluble rocks. We must assume that the liquid which formed the space of dissolution also performed the filling, in fact, that both processes were almost contemporaneous. They must not, however, be confounded with the metamorphic processes, where the idiogenite is expelled, atom by atom, by the xenogenite, for the deposits in spaces of dissolution always show a distinct crustification, hence every single crust must have found a free space waiting for it.

Concerning the origin of spaces of discission, two groups are distinguished—those which do not extend beyond one rock, and where the force which produced them probably has its seat in that rock; and those which extend out of one rock into another, the force which produced them residing outside of the formation.

As to the filling of spaces of discission, they do not represent open spaces of uniform width throughout, for the original fissure was sometimes closed by detritus

formed by the friction of the walls, or by the movement of the country-rock and other causes. Only the places remaining open would permit an active circulation of solutions and a regular deposition from them. When high pressure was present and the rock contained interstices, the liquid doubtless penetrated into the rock, impregnating it with mineral; or a soluble rock was attacked, and spaces of dissolution formed and filled in like manner as the fissure itself. This explains the fact that, in the same vein-plane, rich deposits alternate with poor or barren spots. Where the mineral solutions formed no cavity they must have expelled part of the original material, and to do this they must have had free access, such as might be furnished by original rock-cavities or by secondary cavities.

After the expulsion of the original material the resulting deposit must be massive, showing no crustification. Frequently, however, there are only negative indications of the metamorphosis. It can merely be seen that the deposit is not an original rock, or that it has not been deposited in pre-existing cavities, and hence that it must have been formed by replacement.

There are two kinds of metamorphous-deposits. In the first, the new material has replaced the more soluble ingredients of a heterogeneous rock, and the result resembles an impregnation. In the second, a homogeneous rock has suffered metamorphosis, and the deposit resembles filled cavities of dissolution.

The author then describes the vadose or shallow underground circulation, pointing out how a siphon-action may be set up and the ground-water of one region find a distant outlet.

It is often possible to observe directly, not only the formation but also the filling of these cavities, and thus to obtain valuable material for the explanation of the origin of xenogenites outside the vadose circulation, and not observable in the stages of formation. It is a most important fact that the phenomena of leaching indicate the path of the circulating liquids through soluble rocks, so that we can study the process in its several stages. A good example of the formation of paths of circulation by leaching action is to be found at the Máros Ujvár salt-mines of Transylvania. The filling of open spaces formed by the vadose circulation is important genetically, since it can be directly observed, and furnishes many conclusions applicable to inaccessible subterranean occurrences. The author describes the deposits formed on the walls, roof, and floor of caves, and infers that if circulating liquid fills the whole cavity the attraction of the walls for the precipitated particles is controlling; but that when air or gas enters gravity becomes predominant and draws these particles to the bottom.

In opal and chalcedony-geodes both forms of precipitate can often be seen, the crust uniformly covering the walls and the horizontal deposit.

The ground-water descends in the deep regions through the capillaries of the rocks; at a certain depth it probably moves laterally towards open conduits and ascends through them to the surface. The solvent power of the water increases with temperature and pressure, also with the duration of its underground journeying. Hence, while descending it can dissolve or precipitate only the more soluble substances. But the ascending current in open conduits is undoubtedly loaded more heavily with less soluble substances, which, as the conditions of their solubility gradually disappear in the ascent, must be deposited in the channels themselves.

These ascending currents, heated and often loaded with minerals, are constantly met with in mining. The author here describes the hot springs on the Comstock lode. The quicksilver mines at Sulphur Bank, California, are mentioned as exhibiting the formation of ore-deposits by ascending waters containing large

quantities of sulphuretted hydrogen (H_2S) and the decomposing effect of that gas—oxidized into sulphuric acid (H_2SO_4)—upon the country rock.

The most important factor in ascending waters is carbonic acid, for it is chiefly this gas which in the deep region, under high temperature and pressure, develops a greater solvent power for most of the elements of the rocks.

Minute metallic mixtures are also found in mineral springs, either dissolved in the water itself or deposited at the mouths of springs, where they are concentrated in observable quantities.

The structural features of the deposits of mineral springs, such as crustifications of wall-deposits, pisolitic forms, etc., are identical with those of ore-deposits an additional reason for declaring the latter to have been formed by mineral springs.

The author opposes the views of Dr. Sandberger, who upholds the lateral secretion-theory for the origin of ore-deposits. He thinks that the metals found in the country rock, and stated by Dr. Sandberger to be the source of the deposits, were really xenogenous—derived from the ore-deposit—and is led to this conclusion by the small quantity which can be formed in the country rock.

The various classes of ore-deposits may be divided into four groups.

(1.) Fillings of spaces of discission (fissures, etc.).

(2.) Fillings of spaces of dissolution in soluble rocks.

(3.) Metamorphous deposits in soluble rocks, in simple sediments, in crystallines and eruptives.

(4.) Hysteromorphous deposits (secondary deposits, due to surface-agencies).

The author has attempted to show that in the two regions of underground circulation the formation of ore-deposits has taken place according to different, almost opposite principles. In the vadose region through descension and lateral secretion, in the profound region by ascension, as the product of upward currents. He points out that the original sources of the metallic solutions can scarcely have been reached by mining, and that these sources must lie at still greater depths.

1. *Ore-deposits in Spaces of Discission.*—These may be further subdivided into:—

(*a*) Ore-veins in stratified rocks. Examples of this class, *i.e.*, entirely unconnected with eruptive rocks, are not easily found. The veins of Clausthal in the Harz and Andreasberg are mentioned.

(*b*) Ore-veins in the neighbourhood of eruptive masses. Examples: The Erzgebirge and Pzribram where the rich ore-ground starts from the intersection of the great structural fault with the zone of eruptive rocks, in other words, from the point relatively nearest to the barysphere. The dykes of this district present different kinds of eruptive rocks, and are generally decomposed in the neighbourhood of the ore-veins, no doubt owing to the action of the mineral springs. The stratified rocks—Cambrian sandstone—show near the granites a contact-metamorphosis which has converted them into hornstone.

(*c*) Ore-veins wholly within large eruptive formations. Examples: Hungary, the Dacian gold-field, the Comstock lode. This latter is properly a quartz-vein, in which important ore-concentrations have been formed, not showing any clear crustification (except in the Justice mine). This, however, may have been subsequently obliterated by metamorphosis of the vein-mass, *e.g.*, through the replacement of calcite by quartz, It is also a contact-vein between a diorite foot and a diabase hanging-wall. So far as the sources of the eruptive rocks can be inferred they were all (except the diorite) on the hanging-wall side of the vein, as were also the mineral springs which decomposed these rocks. But the ascending thermal waters encountered in these mines were within the vein itself; whence it

may be concluded that the ore-bearing solutions came by that road from the deep region, and not, according to the lateral-secretion theory, from the side.

The Comstock lode had always been looked upon as an example of the lateral-secretion theory, and this point was freely discussed by members of the American Institute, who seemed disinclined to give up their former opinions.

2. *Ore-deposits in Soluble Rocks.*—Examples: Rodna and Offenbánya in Transylvania, Rézbánya in south-eastern Hungary, Raibl in Carinthia, North of England lead-mines, Leadville, and Red Mountain in Colorado, Nevada, Wisconsin, etc.

3. *Metamorphous Deposits.*—Metamo phism has been defined as the sum of the chemical changes undergone by the sedimentary rocks after their deposition. There are two kinds of metamorphism, general or regional, affecting the rocks over wide areas; and local or contact-metamorphism caused in certain groups of strata by eruptive intrusions. The latter is more important in dealing with our subject. We distinguish the replacement of some constituents of a compound rock, for which the term "impregnation" is used, from the replacement of the whole homogeneous mass by metasomasis.

The metamorphous deposits are reviewed as they occur in (*a*) distinctly stratified rocks; (*b*) soluble precipitates; and (*c*) crystalline schists and eruptive rocks.

(*a*) Metamorphous deposits in distinctly stratified rocks. Some authorities are inclined to believe in the deposition of ores from sea and fresh water. The author has, however, never been able to find genuine ore-beds among rocks deposited by salt or fresh waters, but always only ores of a subsequent introduction, and therefore believes that such ore-beds proper do not exist. Examples: the kupferschiefer of Mansfield, where the clearly defined influence of faults upon the ore-distribution points to a later introduction of ore through the fault-fissures. The copper-bearing veins of Westphalia; copper sandstones of Bohemia and St. Avold; the lead-deposit of Mechernich; Silver Reef in Utah; copper-deposits of New Mexico and Lower California.

(*b*) Metasomatic deposits in soluble rocks. A metasomatic replacement of the original rock-material was clearly proved long ago for some instances, *e.g.*, calamine-deposits, while in other cases, where proof has not been obtained, analogies in the observed circumstances speak for such an origin. Examples: the calamine-deposits of Raibl in Carinthia, Weisloch, Vieille Montagne, and of Laurium in Greece, the limonite-deposits of Cumberland, etc.

(*c*) Deposits in crystalline schists and eruptive rocks. As a general rule, in regional metamorphosis, the older a rock is the more changes it will have been found to have undergone. Yet these changes do not advance in all places uniformly. The action of mineral solutions upon the country-rock of some veins might also be classed as impregnations. As examples of this the author describes some of the Cornish tin capels. The ore-deposits in metamorphous and eruptive rocks occur especially in the great crystalline northern areas, in Scandinavia, Canada, and the north-eastern United States. The zinc-blende mine of Åmmeberg is cited as being of particular interest.

The copper district of Lake Superior offers a number of very interesting occurrences, some of which have not yet been satisfactorily explained. The author is inclined to believe as the cause of the first ore-depositions, the action of hot springs long after the intrusion of the eruptive flows between the sedimentary strata, so that the ores were brought not by or in the eruptives themselves, but by the later springs from great depths and perhaps from considerable distances.

4. *Hysteromorphous (or later-formed) Deposits.*—These comprise:—

(*a*) Chemical effects. On the surface it is especially the oxidizing effect of the atmosphere, the carbonic acid it contains, and the solvent and chloridizing action

of atmospheric precipitation aided by wind and moving water. Example: the limonite-deposit near Rio Tinto. Mine-waters contain the solutions of all substances directly or indirectly dissolved by the vadose circulation, and some of these, encountering suitable precipitants, may be thrown down. These new precipitates will mark the track of the mine-waters. Solutions formed by surface-waters, like those of the mine-waters, mostly find their way to the points where the water-level reaches the surface, yet they may produce impregnations in the deep region by penetrating to greater depths. The decomposition above water-level of gold and silver-bearing deposits facilitates the extraction of these metals.

(*b*) Mechanical effects. Grouped under the head of erosion. Surface-deposits *in situ*. If an ore-deposit, by reason of its greater resistance, crops out above the level of the country, the wind will continually tend to blow away the finer and lighter portions of the detritus formed by chemical weathering; so that in course of time only the heavier portions of the original outcrop will remain. The great agent in the transportation and redeposition of the metallic portions of original deposits has unquestionably been flowing water. With regard to the concentration of these metallic portions, the author believes that the most probable explanation may be based on the principle that the specifically heavier elements of a loose mass are able, with the aid of water, to work their way down through the lighter portions, and thus become concentrated on the bed-rock stream-detritus.

The detrital deposits produced by running water are characterized by the predominance of permeable material, such as sand, gravel, etc. Under this covering mass lies the solid impermeable "bed-rock," and in all the gold-fields of the world the richest pay-deposits are found, as a rule, at the border between the cover and the bed-rock. If the latter is decomposed, fissured, or otherwise loosened, the fine gold will sink into it, making it sometimes rich enough to be mined and concentrated.

Marine detritus. In some regions, the auriferous detritus, after being repeatedly deposited and again swept away, has at last reached the sea. Here the ebb and flow of the tide operate very nearly on the principles of artificial ore-dressing, and one would expect to find a concentration of the heavier particles; but this effect has not been recognized hitherto.

Platinum and tin placers are also given as examples of hysteromorphous deposits.

(*c*) Hysteromorphous deposits of the older geological formations. Examples: Deadwood, Dakota, where the conglomerate of pebbles of quartz, schist, and hæmatite, which lie at the base of the Cambrian series, carry partly coarse gold under such circumstances that there can be no doubt of its secondary origin; the Witwatersrand conglomerate in the Transvaal, which is of Palæozoic age; and in Bohemia, where considerable traces exist of ancient placer-mining in old Permian and Carboniferous conglomerates. A. W. G.

GOLD IN THE HIGHLANDS OF BRAZIL.

By J. C. Branner. *The Engineering and Mining Journal* (*New York*), 1895, *vol. lix., page* 55.

Brazil was at one time the greatest gold-producing country in the world, and it was, until recent years, also the greatest diamond-producing country. The yield of gold and diamonds was at its height while African slaves were still being shipped into the country in large numbers. When the traffic was stopped the price of slaves went up, and many of the mines ceased work for want of cheap labour.

Most of the mining in Brazil for the last forty years has been by *garimpeiros*, i.e., single prospectors washing the pockets of sand and gravel in the beds of streams.

Dr. Eugene Hussak says:—"For more than 150 years gold has been mined in Goyaz, and though mining is almost extinct, being confined to a few hundred *garimpeiros* using the most primitive methods, it is certain that its auriferous wealth is not exhausted."

According to Dr. Arena a cubic metre of rock yields one milreis (4s. 6d.) worth of gold, enough to pay a properly directed hydraulic system employing amalgamation. It is evident that batea-work in such a place cannot pay expenses.

J. W.

A BROMO-CYANIDE PROCESS FOR GOLD EXTRACTION.

By C. A. MULHOLLAND. *The Engineering and Mining Journal (New York)*, 1895, *vol. lix., page* 510.

The bromo-cyanide process is not patented, and is offered to the attention of the mining industry by the author. It differs from the Sulman-Vautin process in that cyanogen bromide is employed by them, whereas free bromine is used by Mr. Mulholland.

The results in most cases were good, and sometimes very good, reaching an extraction of 97 per cent. in a period of time impossible with existing methods.

The following reaction is said to take place:—

$$4\,Au + 8\,KCy + 2\,Br + O_7 + H_2O = 4\,KAuCy_2 + 2\,KBrO_3 + 2\,KOH.$$

The bromine is applied in various ways; the ore is kept agitated, and from time to time the bromine is admitted till the proper volume has been passed into the stuff, or the solution is allowed to percolate through the ore in the presence of the requisite amount of bromine vapour; or, finally, air and bromine in suitable proportion are blown through the ore in the leaching solution. Each of these methods is said to have its advantages.

The bromine used may be recovered again, after the gold has been precipitated with zinc, by concentrating the solution and treating it with hydrochloric acid.

The author specially recommends the process to those who have bodies of tailings, slimes, etc., which hitherto have not been deemed profitable for extraction. The process is said to be simple, rapid, and economical, giving a high percentage extraction, requiring no expensive plant or recondite chemicals, and is considered an advance on the treatment of ores by potassium cyanide solution alone.

J. W.

GOLD-MINING AT CRIPPLE CREEK, COLORADO, U.S.A.

By ARTHUR LAKES. *The Colliery Engineer (Pennsylvania)*, 1895, *vol. xv., pages* 121-123 *and* 145-148, *with ten illustrations.*

The author describes the Cripple Creek mining district, now one of the leading gold-regions of the Western States, dealing with its location, topography, and geology.

Descriptions are given, with drawings, of the Buena Vista, the Victor, Deer Horn, Anaconda, Ida May, Elkton, Moose, and Independence mines.

Professor Lakes concludes by recommending prospectors to search, first, for the "float rock," or broken off portions of the vein-matter, washing by "panning," if necessary; then, if particles of gold are found, trenching should be proceeded with higher up the hill.

J. W.

GOLD EXPORTED FROM WESTERN AUSTRALIA.

Return of gold, the produce of the Colony, entered for export from 1st January, 1886, to 31st December, 1894; showing quantity and value* obtained from respective gold-fields, etc.—*Government Gazette*, January 19th, 1895.

Year.	Kimberley.						Pilbarra.						†Yilgarn.						Ashburton.						Murchison.						Dundas.						Totals.					
	Quantity.			Value.			Quantity.			Value.			Quantity.			Value.			Quantity.			Value.			Quantity.			Value.			Quantity.			Value.			Quantity.			Value.		
	O.	D.	G.	£	s.	d.	O.	D.	G.	£	s.	d.	O.	D.	G.	£	s.	d.	O.	D.	G.	£	s.	d.	O.	D.	G.	£	s.	d.	O.	D.	G.	£	s.	d.	O.	D.	G.	£	s.	d.
1886	302	0	0	1,147	12	0	..			..			..			..			..			..			..			..			..			..			302	0	0	1,147	12	0
1887	4,873	0	0	18,517	8	0	..			..			..			..			..			..			..			..			..			..			4,873	0	0	18,517	8	0
1888	3,493	0	0	13,273	8	0	..			..			..			..			..			..			..			..			..			..			3,493	0	0	13,273	8	0
1889	2,464	0	0	9,363	4	0	11,170	0	0	42,446	0	0	1,858	10	0	7,062	6	0	..			..			..			..			..			..			15,492	10	0	58,871	10	0
1890	4,474	0	0	17,001	4	0	16,055	6	6	61,010	3	9	2,277	0	0	8,652	12	0	..			..			..			..			..			..			22,806	6	6	86,663	19	9
1891	2,699	12	8	10,258	10	10	11,875	0	0	45,125	0	0	12,533	5	23	48,766	10	8	838	14	10	3,187	2	9	2,064	8	16	7,844	16	11	..			..			30,311	1	9	115,182	1	2
1892	1,088	16	22	4,137	12	3	12,892	16	0	48,992	12	10	21,209	9	18	80,596	1	1	0	14	0	2	13	2	24,356	9	12	92,554	12	1	..			..			59,548	6	4	226,283	11	5
1893	1,621	13	23	6,162	9	0	11,698	10	0	44,454	6	0	75,744	10	23	287,829	5	8	447	14	22	1,777	8	8	21,210	8	22	80,599	13	11	147	19	11	562	5	11	110,890	18	5	421,385	9	2
1894	588	12	17	2,236	16	4	16,254	10	0	61,767	2	0	136,828	4	4	519,947	3	9	285	5	10	1,084	0	8	52,946	6	11	201,196	0	7	228	7	12	867	16	6	207,131	6	6	787,093	19	10
Total	21,604	15	22	82,098	4	5	79,946	2	6	303,795	4	7	250,751	0	20	952,853	19	2	1,592	8	18	6,051	5	3	100,577	13	13	382,195	3	6	376	6	23	1,430	2	5	454,848	8	6	1,728,423	19	4

* The Official Valuation is £3 16s. per oz.

† Including for 1894, Coolgardie, viz., 105,329 ozs. 16 dwts. 11 grs., of the value of £400,253 6s. 8d.

M. W. B.

CYANIDING *VERSUS* CHLORINATION FOR GOLD ORES.

Cyaniding or Chlorination. *By* E. A. SCHNEIDER. *The Engineering and Mining Journal* (*New York*), 1895, *vol. lix.*, *page* 461.

The author having had excellent opportunities, recently, of investigating the working of the cyanide and chlorination extraction-methods in the Western States of America, discusses their respective advantages from an unbiassed standpoint.

He is of opinion that under exceptionally favourable conditions the cyanide is cheaper than the chlorination process; but under all circumstances it is safer to operate chlorination-works. Chlorination can be successfully applied to almost any gold ore, whereas the cyanide process gives satisfactory results with only a certain class of ores.

In such localities as Cripple Creek, where the character of the ores is liable to change from day to day, the cyanide process is considered risky; it also requires, in most cases, extremely skilled chemical supervision. On the other hand, the chemical part of the chlorination process is very simple. J. W.

TREATMENT OF ORES IN THE COONEY MINING DISTRICT, NEW MEXICO, U.S.A.

By CARL ANDERSEN. *The Engineering and Mining Journal* (*New York*), 1895, *vol. lix.*, *page* 388.

Mills, where pan amalgamation is used, have been erected at five different mines. The ore, after sorting, has averaged from 6 to 8 dollars in gold, and from 10 to 12 dollars in silver. The extraction varies from 65 to 75 per cent. of assay value, 70 per cent. being an average. The ores are pulverized to 40 mesh, no grinding being done in the pans. From 1·5 to 3 pounds of copper sulphate, and from 10 to 30 pounds of salt are used per ton of ore. The best results have been obtained by treating the pan until steam commenced to escape. The loss of mercury is about ¾ pound per ton, when the charge is carefully settled; but this loss is sometimes greatly increased, by bunches of ore carrying oxide of manganese, which causes the mercury to flour. Great loss has often been caused by millmen not understanding the reaction of the chemicals that they use on the ore treated; of this the author gives a notable instance.

Numerous laboratory experiments, conducted by the author, gave an extraction with cyanide, at 30 mesh, of 90 per cent. of the gold and 80 per cent. of the silver, the loss of potassium cyanide, when using ½ per cent. of lime, being only 1 pound per ton.

The cost of mining and sorting at these mines averages between 4 and 5 dollars per ton. Cost of milling: wages, 2½ dollars; wood, 90 cents; and chemicals, 90 cents; total, 4·30 dollars per ton. J. W.

THE CYANIDE PROCESS FOR GOLD EXTRACTION IN VICTORIA.

The Cyanide Process in Victoria. *By* HENRY BUCKLEY. *The Engineering and Mining Journal* (*New York*), 1895, *vol. lix.*, *page* 582.

The author watched the cyanide process working at the New Golden Mountain mine, Bonnie Doon, Victoria. The plant erected consists of 3 leaching-vats, 18 feet in diameter and 5 feet deep, and 40 tons capacity; also 2 solution-vats, 16 feet in diameter and 5 feet deep, a dissolving-vat, and 2 zinc or precipitating-boxes.

The ore is treated by crushing, dry, in a pair of Cornish rolls, a little lime being added to correct acidity; then through a hopper and Faucett mill, with $\frac{1}{16}$ inch perforated grate, and finally through a pair of Krom rolls into the storage-hopper.

The ground ore being placed in the vat, the cyanide solution is run on. In due time the solution is allowed to pass through the zinc in the precipitating-boxes. When the solution has disappeared under the ore, a washing of water is run on, followed by a second washing, to remove all the cyanide. When the treatment is completed, the zinc is removed and dissolved in sulphuric acid; the residue is washed on a calico filter, dried, heated strongly to oxidize the baser metals, and the resulting mass is smelted in a plumbago crucible with borax and sand, the slags being again treated to free them from adhering beads of gold.

The average percentage of gold extracted by the cyanide was 82·3 per cent. of total contents. The cost of treatment per ton of ore is:—Potassium cyanide, 4s. 7d.; zinc shavings, 2¾d.; lime, 3d.; cost of labour, filling and discharging vats, etc., 5 d.; total, 5s. 6d. J. W.

THE LIMITATIONS OF THE GOLD STAMP-MILL.

By T. A. Rickard. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxiii., pages* 137-147.

The author compares the stamps in use in Colorado with those of California. In the former, the stamp weighs 550 to 600 pounds, falls a height of 18 to 20 inches, and crushes about 1 ton per twenty-four hours. In the latter, the stamp weighs 750 to 850 pounds, falls only 4 to 6 inches, and will crush about 2½ tons per twenty-four hours. Yet each stamp is the best suited to the ore it has to deal with The action of water and bubbles of air in the mortars are fully described, also the method of collecting the fine gold on amalgamated plates.

The author contrasts the methods employed in olden times with those of the present day, and concludes that though the appliances of to-day may show a great advance upon the older and more imperfect type from which they were evolved, yet there is no mining district that possesses a mill which cannot, in some essential, be improved. A. W. G.

IRON PYRITES IN MASSACHUSETTS, U.S.A.

Sulphur Mining. By E. McCormick. *The Colliery Engineer* (*Pennsylvania*), 1895, *vol. xv., pages* 243-244.

In the western parts of Massachusetts is the largest deposit of pyrites known in the United States. The mining town called Davis has sprung up around these workings. The mines were first opened in 1881. The Davis Sulphur Ore Co. are the only producers in the State; their pyrites contains from 50 to 51 per cent. of sulphur, and is said to be quite free from arsenic. An analysis of the ore shows:—

Sulphur, 51; iron, 45·75; copper, 1·5; and silica, 1·7 per cent.

The vein varies in thickness from 12 to 45 feet, and the lower workings are said to yield the same grade of ore as did the first opening of the mine.

In mining, a shaft 60 feet deep is sunk, a pillar 20 by 60 feet being left; a drift is driven along the vein to the next shaft, and a shaft sunk again to the next level.

The ore is screened and divided into lump ore and wet smalls. The pyrites is extensively used for the manufacture of sulphuric acid. J. W.

A NEW IRONWORKS IN THE URAL.

Ein neues Hüttenwerk im Ural. *By* A. MESTCHERSKY. *Stahl und Eisen*, 1895, *page* 483.

One of the consequences of the building of the great Siberian railway has been the starting of a number of undertakings for the manufacture of iron, among them the rail-rolling mills of Bogoslowsk in the Ural (province of Perm), owned by Mrs. N. M. Polowzeff. The district contains a quantity of iron-ore, and the large forest on the same estate ensures a plentiful supply of fuel for many years, while easy communication with the interior of Siberia exists *vid* the navigable rivers tributary to the Irtish and Obi.

For treating the ore—which is a red ironstone containing 90 per cent. of ferric oxide (Fe_2O_3), about 3½ per cent. of silica (SiO_2), ½ per cent. of lime and magnesia, and 0·04 per cent. of phosphorus—there are four charcoal smelting-furnaces of 4,500 cubic feet (125 cubic metres) capacity, equal to a daily output of 30 to 35 tons of pig iron, fitted with six Cowper blast-heaters. Four 15-tons Siemens-Martin furnaces, with wood and turf generators, travelling steam-cranes, hydraulic lifters and cranes for loading the ingots into trucks, are provided for the production of steel.

The ingots are of 1½ tons (1,500 kilogrammes) each, rolled into six-rail lengths after re-heating in the 4 rolling-furnaces; all the operations connected with the transfer of the ingots, the adjustment of the rollers, etc., are accomplished by hydraulic power. After the first rolling and cutting in half by the shears, each of the halves is reheated in a Siemens furnace and rolled out to a 3-rail length in a triple Erdmann rolling-mill with rollers 30 inches (750 millimetres) in diameter. The engine driving this mill has a cylinder 50 inches (1·25 metres) in diameter, 5 feet (1·5 metres) stroke, and the fly-wheel weighs 60 tons. Two pendulum saws, 3 double straightening presses, and 4 milling machines are employed in the finishing process. There are sixteen boilers (water-tube) fed from a water-tower supplied from the river, some 1,500 feet away, by a pair of pumps raising 132,000 gallons (600 cubic metres) per diem.

For the supply of fire-resisting materials a factory calculated to produce about 1,000,000 bricks a year has been erected, and there is also in contemplation the establishment of a foundry with 3 cupola-furnaces, 4 revolving-cranes, and 1 travelling-crane, together with a number of special and ordinary tool-making machines.

Some 3,000 men were employed in erecting the works, which was done solely by Russian labour, but nearly all the machinery was supplied by German or French firms. With the 120 houses for married and 20 barracks for unmarried labourers, and the dwellings for the officials, the new establishment forms quite a little town. In view of its isolated position a year's stock of corn and forage has to be kept in hand. C. S.

THE MINING LAWS OF TURKEY.

La Turquie, Constantinople, 1888.

[The following digest of the mining laws of Turkey is made from a French translation of the mining Act, which appeared in *La Turquie*, of Constantinople, in 1888. The requirements of the law are, however, very loosely observed in practice. Reform is urgently needed, as the country has splendid mineral resources, and their development would add enormously to the revenues of the country, which are by no means in a flourishing state. What is most required at the present time is a

proper survey of the country, and the enforcement of the regulations for obtaining a *firman* or title without unnecessary delay or expense. The foreigner at present has, in spite of the law, endless trouble and difficulty in obtaining a *firman*, and, as a rule, only succeeds by the interest of high officials and a large expenditure of *backsheesh*. Even when the *firman* has been obtained he is very likely to be involved in litigation on the subject of boundaries, owing to the absence of reliable maps.—EDITOR.]

Chapter 1 (*Arts.* 1-3).—*Classification of Mines.*

For the purpose of this Act mines are divided into three classes :—

1. Mines with underground workings (mines).
2. Surface mines (minières), embracing iron ore deposits, turf mines, etc.
3. Quarries (carrières). These do not come under the present Act, but have their own special code of regulations.

Chapter 2 (*Arts.* 4-8).—*Generalities.*

No mine may be worked without a *permis* or *iradé impérial* (*firman*). The term of a concession is usually 99 years, but mass mines, such as chromite, emery, boracite, etc., may have a longer or shorter concession, but never less than for 40 years. The plant, machinery, horses working underground, instruments, etc., cannot be seized for debt, or the stores of one year ; but the ore at grass and the shares or profits of a company can be taken. On freehold land (*mulk*) and on endowed lands, *i.e.*, lands belonging to the mosques (*vakouf*), mines may be worked ; but one-fifth of the dues only go to the mining administration, the other four-fifths going to the proprietor of the *mulk* or *vakouf* land.

Chapter 3 (*Arts.* 9-22).—*Discovery of Mines.*

This chapter provides for mining exploration. The owner of the soil can mine without special permission, but must give notice to the administration of his intention, and must say what the mineral is. On public lands permission must first be obtained from the Government. Permission will not be given to mine on land near towns where the workings would interfere with the interests of the inhabitants. No one can explore enclosed land without permission of the surface-owner, but if permission is refused, application may be made to the governor-general of the province, with full particulars of the mine and of the land, and an undertaking to make good all damage. If this petition is made by a company, such company must be one recognized by the Government. The petition will be registered and examined by the administrative council of the province. If approved, permission to mine may be granted for one year, renewable for a second year. The permission (*permis*) may be sold, but no mineral can be sold without the express permission of the mining administration. Anyone breaking this rule is liable to a fine of from £*t*25 to £*t*100 and loss of the *permis*, and double dues will be levied on any ore sold. The holder of a *permis* may send a trial shipment of the ore to Europe once only, but the weight must not exceed 100 tons. Dues must be paid according to the nature of the ore, the value being calculated from the price-lists of Europe, and deductions being allowed for cost of freight.

Chapter 4 (*Arts.* 23-39).—*Conditions of Holding and Working Mining Concessions.*

This chapter deals with the regulations for obtaining and holding a concession (*firman*), and for working the same. Concessions may be held by foreigners as well as Ottoman subjects. No *firman* will be granted till the following points have

been settled:—(1) That the mine discovered can be worked; (2) that the working of the mine will not interfere with the working of an adjoining mine; (3) that the mining works will not interfere with any fortifications.

In the event of the first discoverer having failed to comply with the regulations laid down for acquisition of a *permis*, the mining administration may grant the concession to another; but compensation will be granted to the first discoverer according to the work done, and a maximum indemnity of half a Turkish *lira* per day, and 5 per cent. of the net produce of the mine. The amount of the indemnity is to be recorded on the *firman* of the concessionaire, and until it is paid the mine will be considered mortgaged to the first discoverer or original holder of the *permis*. Objection to the amount of the indemnity may be made before the Council of State, but the settlement of the amount need not stop the granting of the *firman*.

The holder of a *permis* or right to prospect, wishing for a concession or *firman*, must make a petition for the same to the mining administration. This petition must contain full particulars of the mine, and must state what indemnity will be paid to surface-owners. The petition must be accompained by a plan (in three copies) to the scale of 1 : 500, an engineer's report, and a sample of the ore or mineral. The mining administration will examine it, and, if in order, will place it on its register. If not in order it will be sent back to the concessionaire for correction.

If it be accepted, the particulars of the concession must be published at the expense of the concessionaire in a Turkish and a French newspaper for two months; also on posters, and in the official journal of the *vilayet* or province. At the end of two months the governor-general of the province shall report to the mining administration whether or not objection has been raised.

All documents sent to the mining administration shall be examined by the technical department, and a report made. Then six months after the report has been made the administration can proceed to obtain a *firman* from the Sublime Porte. The *firman* will bear the signature of the Sultan. The *firman* must then be published as before, but once only.

Concessionaires may not amalgamate their concession with others without permission of the Government.

The *firman* only recognizes the mineral or minerals mentioned in the petition, and the right to mine other minerals discovered later must be obtained by a fresh *firman*. The original concessionaire has always the right to any new discovery, subject to certain recompenses being paid to the discoverer. The penalty for working a mine without a *firman*, or for extracting more than a trial 100 tons, is from £*t*100 to £*t*1,000.

The mines belonging to the State come under the same regulations as private mines, as regards boundaries, indemnities, and working conditions.

The mining administration has to publish an official list of mines abandoned, mines discovered, and mines whose concessions have been confiscated. It is authorized to lease mines to the highest bidder.

The law only recognizes mines which are registered, except those that have been worked for 99 years since the date of the present law. All others are unknown, and can be denounced.

Chapter 5 (Arts. 40-64). – Rights and Obligation of Concessionaires.

On delivery of the *firman* a payment of from £*t*50 to £*t*200 has to be made. The dues payable by a concessionaire are as follows:—(*a*) According to area of concession (so much per *djérib*); (*b*) according to the production. The area tax is always payable, whether the mine is worked or not, as long as the *firman* is held.

It amounts to 10 piastres (1s. 8d.) per *djérib*. On the *mulk* land and *Evkaf-sahibi* lands the payment is made to the proprietors of the soil, on common land to the Government.

The production tax varies between 1 to 5 per cent. of the value at mines in the case of copper, silver-lead, coal, and other deep mines, and 10 to 20 per cent. in the case of emery, boracite, meerschaum, and other mass mines.

Proper returns have to be made on March 1st in each year to the governor of the province. The value of the ore is arranged according to a tariff (revised every six months), and the percentage payable is fixed according to the richness of the mine by the mining administration, and such percentage is entered on the *firman*. Minerals for export are weighed at the port, and minerals treated at home at the smelting or reduction-works. The penalty for sending incorrect returns is from £*t*5 to £*t*25, and double duty on the amount not reported.

The governor-general may authorize the postponement of payment of dues for a term of two months. If not then paid, the mine may be sold, in accordance with the conditions laid down in chapter 2; and if the sale does not cover the amount of dues, the private property of the concessionaire may be seized. On completion of the *firman*, the limits of the concession are laid down on the mine plan by an engineer appointed by the mining administration. An official report is drawn up, and three copies made, one for the mines office, one for the local authorities, and one for the concessionaire.

The concessionaire must commence to work the mine within two years of the delivery of the *firman*. If he fails to do so, the concessionaire is asked in writing the cause of his not doing so. If the cause is not considered valid by the mining administration, a further six months is allowed the concessionaire, and if he then fails to work the mine the concession becomes forfeited. Right of appeal, within three months, to the Council of State is allowed. If the Council of State refuse the appeal, the withdrawal of the *firman* is publicly advertized in three newspapers of Constantinople, and in the local press.

Then follow regulations regarding the working of the mines. A capable manager must be appointed. Surface and underground plans, of a scale of 1 : 500, must be kept, under a penalty of £*t*5 to £*t*25. Mines must be inspected by Government engineers.

The mining administration settle all technical disputes between neighbouring mines. If a mineowner desires to take private property within the limits of the concession he may take it, provided he pays an indemnity of twice the value of the crop that such land would produce, and must return the land in the same condition as he found it. Concessionaires may purchase private land for mining purposes; if no agreement is come to, the concessionaire must pay twice the value assessed by the Government. With the exception of the engineer and superintendent, the employés must be all Turkish subjects. For a first offence against this regulation a fine of £*t*100 is imposed; on a second offence the concession may be confiscated.

Chapter 6 (*Arts.* 65-71).—*Inspection of Mines.*

Each *vilayet*, or province, is placed under the charge of a Government engineer, whose duty it is to see that the regulations are observed and the mines properly worked. The concessionaire is obliged to keep a medical store and a qualified doctor. Concessionaires are obliged to pay an indemnity to all those who meet with an accident or to their families. The amount of the indemnity is fixed by a tribunal. Where the accident was preventable, the concessionaire is liable to a fine of from £*t*50 to £*t*100.

Chapter 7 (*Arts.* 72-77).—*Abandonment of a Mine.*

No mine can be abandoned without the consent of the mining administration. Six months' notice must be given. The concessionaire must pay all dues to date, and must sell the plant and machinery to a new concessionaire or to the State at a valuation if required to do so.

Chapter 8 (*Art.* 78).—*Concessions granted previous to the present law.*

Such concessions are controlled by the present regulations. The dues, however, both fixed and unfixed, shall be maintained on the old tariff for ten years, after which time all mines shall come under the tariff of this Act.

Chapter 9 (*Arts.* 79-92).—*On Surface Mines.*

All surface-mines may be worked in perpetuity by the surface-owner, but only after permission has been obtained by *firman*, which costs £*4*. If the surface owner of a surface-mine either does not work it himself, or abandons it after a short working, the surface-mine may be given to another person, who must pay double the value of the land and full value of the buildings, etc. All surface-mines, except on *mulk* or *vakouf* land, come under the jurisdiction of *terrains domaniaux*. During the working of the mine they shall be free from all taxes except those on production.

The demand for permission to work a surface-mine must contain full particulars as regards name of concessionaire, situation of mine, mineral worked, etc., and a plan on the scale of 1 : 500. The permission shall determine the limits and the methods of working with due regard for the security of buildings, roads, water-courses, etc., and shall fix the indemnity to be paid. It shall decide as to what water may be taken for washing the ore. Surface-mines shall be under the control of Government engineers, like deep mines. A surface-mine may not be changed into a deep mine (*i.e.*, no underground workings may be made) without a special *firman*. The concessionaire may establish smelting-works with the permission of the mining administration.

Disputes of title are decided by the administration; other disputes by the ordinary courts of justice in which, in the case of foreigners, questions of title are also decided. W. F. W.

MEERSCHAUM MINING IN TURKEY.

ANON. *The Engineering and Mining Journal* (*New York*), 1895, *vol. lix.*, *page* 464.

Rich deposits of meerschaum are found 20 miles to the south-east of Eski-Shehir, a station of the Anatolian railway. Pits from 25 feet to 120 feet deep are dug, and when the bed is struck, horizontal galleries are made; but more than two galleries are seldom to be found in one pit.

The stone as extracted is called *ham-tash* (rough block), and is soft enough to be easily cut with a knife. It is white, with a yellowish tint, and is covered with a thickness of about an inch of red clayey soil. In this state the blocks are purchased by dealers, by the bulk, the price varying from £5 to £30 per load, according to quality. Most of the finest specimens of the dried meerschaum are sent direct to Paris. It is maintained, locally, that the Eski-Shehir meerschaum is superior to that of the Crimea and also to that of Greece. J. W.

THE FLOURING AND SICKENING OF MERCURY.

By T. A. RICKARD. *The Engineering and Mining Journal (New York)*, 1895, *vol. lix., pages* 460-461.

The author thinks that there is often misconception as to the meaning of the above terms; in the same way, the reactions which produce the sickening of mercury are but rarely, and then only vaguely, understood. In order to aid all in getting their ideas on the subject clarified, the author discusses at length the causes of the flouring and sickening of mercury which have come under his observation in practical work. In conclusion, he defines flouring as the minute subdivision of mercury by mechanical causes, and sickening as the rendering of such a condition permanent by the intervention of a coating of some foreign substance, effectually preventing the coalescence or reunion of the globules previously formed.

J. W.

THE HEAT-TREATMENT OF STEEL.

By H. M. HOWE. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxiii., pages* 466-541.

This paper may be regarded as a report of progress of an investigation into the influence of prior exposure to different high temperatures, under different conditions, on the properties of steel after it has been cooled completely.

Experiments on Toughening Manganese-Steel by Sudden Cooling.—The result of these experiments is given in a series of tables. The author is opposed to very violent quenching as liable to induce injurious stress. It is best to quench from a very high temperature. Slightly deferring the immersion after heating is practically harmless. Slightly retarding the cooling does but slight harm. Castings should be heated slowly. Quenching effaces the injury done by previous slow cooling. The influence of repeated quenching depends on the maximum and minimum temperatures to which the manganese-steel is subjected, also on the time taken in the reheating and cooling.

The author then turns to the retardation-points of carbon-steel, *i.e.*, the points at which the regularity of the rise and fall of temperature during the heating and cooling is broken; these points are clearly shown in graphic tables, and are called the W-change and V-change respectively.

It appears that the V-change and the W-change are the opposite phases of the same transformation, the W-change conferring the hardening property, the V-change removing it. The marked retardation which occurs at W, and which shows that some deep-seated change is occurring, does not take place unless the metal, since last rising past W, has been so far cooled as to induce the V-change, and *vice-versâ*. Neither change is instantaneous.

Prof. Osmond's theory was that sudden cooling hardens steel, not by retaining its carbon in a special hardness-giving condition, but by retaining the iron itself in a purely hypothetical, special, hard, allotropic state of B-iron. The author is not of this opinion.

The results of a series of experiments on rail, hard and soft tool, soft basic, and other steels are given, showing the influence of slow cooling from high temperatures. Briefly, heating to W has in some way a wonderful toughening effect, which is lessened if the temperature rise even very slightly above W.

A description of the double-tube-muffle furnace, with the aid of which these experiments were carried out, and some notes on thermo-electric pyrometry, conclude the paper.

A. W. G.

THE MICROSTRUCTURE OF INGOT IRON IN CAST-INGOTS.

By A. MARTENS. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxiii.*, *pages* 37-63.

The author describes the method employed in examining the microstructure of the eight samples submitted to him, three of which were taken from Bessemer ingots, two from Thomas (basic Bessemer) ingots, and three from Martin (open hearth) ingots.

The hardness of the samples varied from soft to very hard, and the proportion of carbon from 0·06 to 0·50 per cent.

The paper is beautifully illustrated by reproductions of microphotographs of the sections taken by the author, and a description is given of each.

A. W. G.

SULPHUR IN CAST-IRON.

By W. J. KEEP. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxiii.*, *pages* 382-399.

The object of this paper is to dispel the superstition regarding the influence of sulphur in the fuel upon the castings produced. The author describes in detail various tests which led him to the following conclusions: that percentages of sulphur that could be retained by grey cast-iron cannot materially injure the iron, except through the increase of shrinkage. Again: the testimony is, that most of the sulphur contained in the pig-iron is lost in re-melting, and that it is impossible for carbonized iron to absorb any damaging amount of sulphur from the fuel. The higher the carbon, or the higher the silicon, the smaller will be the influence exerted by sulphur. The influence of sulphur on all cast-iron is to drive out carbon and silicon, and to increase chill; to increase shrinkage, and, as a general rule, to decrease strength. But if, in practice, sulphur will not enter such iron, we shall not have any cause to fear this tendency.

In every-day work it is sometimes found that iron which was grey when put in the cupola comes out white, with increased shrinkage and chill, and often with decreased strength. This is caused by decreased silicon, and can be remedied by an increase of silicon.

A. W. G.

A NEW PROCESS FOR THE PRODUCTION OF PIG-IRON, REFINED IRON, INGOT-METAL, AND WELD-METAL.

By A. SATTMANN *and* A. HOMATSCH. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxiii.*, *pages* 3-37.

In this process a great saving is claimed by the substitution of gaseous for solid fuel, which would be more marked in localities where suitable blast-furnace fuel does not occur near the iron ores, so that in the manufacture of pig-iron heavy transportation-charges have to be levied on the product, thus raising the cost of production very appreciably.

Briefly stated the process consists of four phases:—

1.—Heating of the ore and flux, and, if necessary, calcining the ore with gas fuel.

2.—Reduction of the ore and carburization of the resulting iron-sponge with reducing gas.

3.—Melting of the carburized iron-sponge by the direct combustion of solid fuel.

4.—Separation of the resulting pig-iron from the slag, and refining with oxidizing gases immediately after the melting, thereby producing a more or less carburized metal.

The paper is illustrated with drawings of various furnaces for intermittent and continuous working, and concludes with tables of heat-calculations and a thermal balance-sheet. A. W. G.

GENERAL AND SPECIAL OBSERVATIONS CONCERNING ORE-DRESSING.

By O. BILHARZ. *Transactions of the American Institute of Mining Engineers, 1893, vol. xxii., pages* 225-236.

Although the rules laid down by Mr. Rittinger in 1867 concerning the designing of dressing-works still possess authority, some modifications, suggested by practice, may be of value at the present time. The general aim in mechanical preparation should be to secure the highest results with the simplest and yet most perfect means and arrangements, so far as those desiderata can be practically combined.

Very fine crushing should be avoided on general principles, since loss is inevitable in the treatment of slimes. Liberation of even finely disseminated minerals should proceed by successive steps.

For classification, the syphon-classifier, in its improved form, may be pronounced the most perfect apparatus. The discharge-opening consists of a slit the entire breadth of the channel, so that variation of speed in the discharge is avoided. For slimes and fine sands no objections can be offered to the use of spitzkasten.

Classification according to size of grain has been recently pronounced superfluous by Mr. Hoppe, and the English practice of jigging without preliminary sizing has received general acceptance. Where the substances differ widely in specific gravity a satisfactory result can always be achieved without careful sizing; but in materials possessing very slight differences of specific gravity, a very accurate classification by grain is necessary.

In choosing a site for the dressing-works, the question of transportation should be the main factor. Repeated handling or tipping should be avoided, and the best utilization of suitable differences of level is to be studied. The several machines should be relatively so arranged that the middlings of each can be carried forward in the shortest and simplest way to the next following manipulation. In designing modern works special weight must be laid on the securing of as much lighting surface as possible. Especially to be recommended is the use of overhead lighting.

Works for mechanical preparation may be divided into: (1) Works treating coal and bituminous substances; (2) those treating metallic substances.

The latter may be subdivided according as they perform a simple or multiplex separation, *i.e.*, the separation of several constituents.

(1) *Works for the Preparation of Coal and Bituminous Substances.*—These operate on large quantities, and the following are controlling conditions:—(*a*) Convenient delivery of crude material; (*b*) maximum simplicity and rapidity of its treatment; (*c*) maximum facility for disposing of both the valuable product and the refuse.

Coal with abundant lumps does not, as a rule, require an extensive plant, unless the coal is largely intercalated with shale. Especial weight is laid upon the production of separate sizes to meet commercial demands, hence screens have become one of the most essential features of coal preparation. Essentially, therefore, an

establishment of this kind will consist of (1) coarse screens for lumps; (2) sizing-apparatus, drums, sieves; (3) coarse jigs; (4) classifiers for small; and (5) fine jigs (felspar-washers).

(2) *Works for the Preparation of Metallic Substances.*—With simple separations, the arrangements are similar to those of a coal-washing plant. In the preparation of ores of complex composition, the material is generally tipped over screens to separate the large from the small. The former, mostly rough angular pieces, are broken in rock-breakers, and a portion goes to the hand-sorting tables, while the smaller pieces, together with the smalls from the mine, are delivered to rock-jigs. Jigging gives concentrates and middle products, the latter being crushed or stamped, followed by classification and fine jigging.

With ores containing free gold, reduction in stamp-mills to fine sand and flour is generally necessary. The escaping pulp passes first over amalgamating-plates, thence to the amalgamators.

With ores containing combined gold, the treatment is directed to the separation of the ingredients which carry the gold. The crushing must be performed with great care, and by successive stages. All tailings must be carefully saved, since they may be further treated by a chemical process, such as the MacArthur-Forrest. Successive crushing and concentration by jigging is recommended for ores of this class. H. W. H.

THE BARTSCH PERCUSSION-BUDDLE.

Die Aufbereitung der Schlämme, insbesondere mit dem Bartsch'schen Stossrundherd. *By* JULIUS SCHWARTZ. *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1893, *vol. xli.*, *pages* 612-616.

The advantages of the Bartsch round percussion buddle are a conical surface combined with the equally necessary percussion-motion, a rotatory system of delivering the ore, a peculiarly curved sieve, continuous work, simplicity and efficiency. It consists of a conical iron plate or surface, resting on pulleys secured to supports below, and fixed in the centre to a bracket upon which they rotate. The percussion action is imparted by a cam which fits into a notch in the plate, and pushes it round; the notch and plate are forced back into position by two springs and driven against buffers. These impulses are rapidly repeated, and being given at the outer edge of the plate, they increase in energy from the centre to the periphery. A roller on the notch diminishes the friction of the cam, and makes the machinery work easily.

The cam-shaft drives simultaneously a worm-shaft which sets the button of the plate in rotating motion. By arranging all the driving-gear together below the plate, space is economized, and the machine is independent of all adjuncts.

The tailings are delivered to the buddle through a funnel and charging-shoot, from whence they are equally distributed over the surface of the plate. The light and worthless part of the slime is carried off at once, while the heavier minerals are deposited by the radial motion of the plate, rinsed through a curved sieve, the current of the water increasing towards the periphery of the sieve, and separated according to their specific gravities. The speed of the waste slime is only half as great when it leaves the plate as at first, and hence it is easier to separate. The intermitting thrusts are in the reverse direction to the rotating motion of the sieve, and this causes not only the deposition of the tailings but also a radial displacement of the slime, the heavier particles in it being first pushed towards the sieve. If the tailings are very rich in ore a second sieve is used. The quantity of water for the sieve is regulated by a float. The rate of delivery of the slime, the

speed of the sieve and of the stream of water, and the number and violence of the percussion movements can be varied to any extent while the apparatus is at work.

The machine only occupies 172 square feet, and can be driven by ¼ horse-power. The fall required for the delivery of the slime is 5½ feet.

In the Lohmannsfeld mining works at Altenseelbach in Westphalia, five Bartsch buddles have been erected, and experiments have proved their efficiency. The ore here treated consists principally of iron spar, with lead and sulphate of zinc, copper ore and ullmanite, the strata being composed of greywacke and quartz. The tailings are very fine mixed slime with 5·53 per cent. of lead, 8·80 per cent. of zinc, and 16 per cent. of iron, and after washing they yield—

	Lead. Per Cent.	Zinc. Per Cent.
Lead ore	68·8 ...	3·15
Spathic blende	1·87 ...	16·69
Residual products	1·97 ...	9·30

At present from 9,900 to 11,000 pounds of tailings (dry weight) are dressed in 10 hours. More mineral and of better quality would be obtained if the ore were less difficult to dress.

In another mine in Germany with a similar apparatus and tailings of about the same quality, but more carefully sorted, the yield of pure lead was 77·36 per cent.

In a gold-producing mine at Kremnitz, Hungary, of which the author has the management, a Bartsch percussion-buddle was introduced experimentally in September, 1893, and with excellent results. Being found superior in every respect to the Frue-vanning machine formerly used in this mine, 3 more Bartsch machines were set up in the course of the year. From 10 to 12 tons of fine tailings are dressed in this buddle in 24 hours. There are 80 thrusts, and 20 revolutions of the sieve per minute, and the latter requires about 12 gallons of water per minute. With gold and silver-bearing pyrites, nearly 97 per cent. may be extracted by this machine, however rich the ore in these metals. The intermediate products are about three times as much as the clean slick. The weight of the buddle as made in Austria is about 5½ tons, and the price about £200.

B. D.

INTRODUCTION AND DEVELOPMENT OF MAGNETIC SEPARATION OF IRON ORE.

By Axel Sahlin. *The Engineering and Mining Journal (New York)*, 1892, *vol. liii., pages* 616-617, 638-641, *and pages* 662-664, with 15 *figures.*

The author treats the subject under the following headings:—Part I., the importance and use of concentrates; Part II., preparation of the ore—(*a*) mining, (*b*) the drying process, (*c*) reduction; Part III., methods of separation, describing the following separators, viz., the Conkling, the Buchanan, Edison, Wenstrom, Ball-Norton, Hoffmann, Lovett-Finney, and the Chase.

Drawings are given illustrating the Krom crusher, the Gates, and the Sturtevant mills, also of most of the separators above-mentioned. Magnetic separators have now been used, not only to concentrate magnetite, but to separate it from more valuable ores, or from apatite, etc., which injuriously affect the iron produced at a later metallurgical stage.

The process was used successfully to separate magnetite from fine blende at Pzribram, Bohemia, and it is also used in the south of Europe for this purpose. In Australia it is used for separating auriferous bismuth ores from magnetite.

It seems probable that magnetic separation will have further applications in the future.

J. W.

CONCENTRATION OF MAGNETIC IRON-ORE AT WELDON, NEW JERSEY, U.S.A.

By AXEL SAHLIN. *The Engineering and Mining Journal (New York), 1893, vol. lii., page 588, with 2 figures.*

The Weldon mine produces about 250 tons of magnetic ore per day. The vein mined varies from 5 to 10 feet in width, and the ore averages about 40 per cent. of iron with about 0·40 per cent. of phosphorus. At bank the ore is assorted, and all lumps not purely magnetic are sent to the concentrating plant, while the clean ore, averaging about 62 per cent. of iron and amounting to about 50 per cent. of the total output, is put on cars and shipped, chiefly to the ironworks at Swedeland, Montgomery County, Pennsylvania.

The low-grade ore averages from 0·75 to 1 per cent. of phosphorus, and about 30 to 35 per cent. of iron. This ore is passed through a jaw-crusher which reduces it to about 1½ inches size, thence it goes through one pair of very efficient 22 by 15 inches Krom rolls to an 8 meshes revolving screen. Rejections from this screen are returned to the rolls.

The ore thus reduced is now conveyed to a Lovett-Finney magnetic separator wheel which consists of a shaft, on which are placed two 30 inches iron discs, 50 inches apart. The space between the discs is wound with No. 14 insulated copper wire, forming a solenoid. One end of the shaft is hollow, and through this central aperture are passed the ends of the wire which connect with the commutator attached to the shaft. From the rim of each disc extend, alternately, a number of iron bars, each bar extending almost to the edge of the opposite disc, but insulated from this as well as from the adjacent bars. The spaces between the bars are filled with non-conducting cement, giving to the finished wheel the shape of a solid cylinder, 30 inches in diameter and 50 inches long. Over this cylinder travels an endless belt of ordinary canvas, held tight by an adjustable pulley. An apron of copper is placed under the magnetic wheel, closely following the curvature of the same. Over the apron the crushed ore is carried by a liberal flow of water. An electric current of 13½ ampères and 110 volts is run through the wire of the wheel, which is revolved at the speed of 14 revolutions per minute. The discs and bars being thus magnetized, the magnetic particles of ore are attracted to the wheel, and attach themselves to the endless belt. The non-magnetic particles are in the meantime washed off and carried away by the water. When the belt leaves the top of the magnetic wheel it carries with it the collected concentrates, which are shed into a water-tank, through which the belt passes before returning to the separator. From this tank the concentrates are lifted by a flight-conveyor, and deposited directly on the railroad car ready for shipment.

To operate the concentrating plant, above described, 2 men and 1 boy are sufficient. The dynamo requires 3 horse-power, and the crushing machinery and magnetic wheel 30 horse-power, a total power of 33 horse-power.

The concentrates had, during the past six months, averaged about 67 per cent. of iron, while the tailings had not run above 8 per cent. J. W.

MAGNETIC CONCENTRATION OF IRON-ORE AT THE CROTON MAGNETIC IRON MINES, BREWSTER, NEW YORK, U.S.A.

By W. H. HOFFMAN. *Transactions of the American Institute of Mining Engineers, 1891, vol. xx., pages 602-609.*

In 1888, the author was engaged by the lessee of the above mines to adapt an old water-jigging mill to the magnetic process. The ore to be treated consists of compact, moderately fine-grained magnetite in a gangue composed mainly of quartz

and hornblende, besides felspar, apatite, and mica, and more or less pyrites and pyrrhotite. The grain of the ore permits a fair disintegration of its constituents by crushing it to pass a 12 meshes screen (a finer mineralization would increase the cost of concentration).

The ore has been examined by Messrs. E. McMillan and F. W. Gordon and Prof. N. S. Shaler, who state that the average amount of metallic iron in the ore beds ranges from 37 to 42 per cent., sulphur averages from 1·7 to 2·2 per cent., and phosphorus from 0·07 to 0·43 per cent. Since May, 1891, they have been roasting and concentrating this material to 68 per cent. in metallic iron, 0·44 per cent. in sulphur, and 0·036 per cent. in phosphorus. Previous to this, the concentration ran about 66 per cent. of iron.

The Croton mines have produced and sold at a fair profit from 50 to 220 tons of concentrates per day during the years 1890-91, the mill having only been at work 20 months. Commercial success in concentrating any ore includes economical mining, preparation, and separation; but in the Croton ore the presence of sulphur calls for very economical roasting, as well as economy in all the other processes.

The cheap roasting of the ore of the Croton mine, which contains about 2 per cent. of sulphur, was really one of the first problems encountered in reclaiming that property, and was really as important a factor as the concentration.

The best size of the ore for economical roasting was found to be such as would pass through a $2\frac{3}{4}$ inches ring, and give the most rapid work for the quantity of fuel consumed. Crude Lima oil is used for roasting, 3·6 gallons being used per ton of raw ore. The average temperature of the roasting is 1,250 degs. Fahr. Davis-Colbey roasters, remodelled to burn oil, are used, and have done excellent service. Hoffman roasters are also in use.

The ore is conveyed automatically from the roasters to the Sturtevant mills, where it is ground to 12 meshes size, all coarser material from the screens being returned to these mills by elevators. Barring the numerous experiments with various types of magnetic separators, the experiments in crushing have been most elaborate. Nearly all the best known methods of grinding ores have been tried at the Croton mines during the past 3 years. The Sturtevant mill has proved superior to any other machine for grinding iron ores. The screen-block openings in the Sturtevant mills are $\frac{1}{4}$ inch wide, and the coarsest material passing through them is less than $\frac{7}{32}$ inch thick, while the finest material would be rejected by a 60 meshes screen. After cooling to 350 degs. Fahr., the ore is quite friable as it enters the Sturtevant mills, and there is no difficulty in grinding 22 tons per hour with the 20 inches mill, and 16 tons in the same time with the 15 inches mill. One set of Sturtevant mill-bushings will grind from 4,000 to 6,000 tons of ore (according to the depth of the chill in the bushing), the cost of each set being 16 dollars. The screen-blocks for this amount of ore cost 9 dollars. This is less than one-half the cost of renewal on any other machine formerly used at this mill. At 22 tons per hour the 20 inches mill requires 94 horse-power to drive it, and the 15 inches mill requires 70 horse-power. The ground ore from the Sturtevant mills is elevated to screens, covered with slotted steel plates.

The screens deliver their finished product to two bins, placed on the floor above the separating department, each having a capacity of 80 tons. Eight chutes deliver the ore to the separators, nine in number. The separators were designed and constructed by Messrs. W. D. Hoffman and T. Blass at the Croton mines. On eight of the machines, the ore is passed before the magnets twice to bring the loss in tailings to 8 per cent. Mr. W. D. Hoffman has lately designed two new separators, one of which is giving, with one pass, concentrates of 68 to 70 per cent., with a loss of 6½ per cent. of iron in the tailings. The cost of separation is about 3½d. (7 cents.

per gross ton (2,240 pounds) of concentrates, and this includes all repairs to separators. Repairs and supplies throughout the milling plant amount to 0·9d. (1·8 cents) per ton of raw ore ground.

During the previous winter and spring a continuous run of five months was made of 20 hours each day, and the average cost of a gross ton of concentrates for the whole term was 8s. 9d. (2·10 dollars). Improvements had reduced this amount to the figure of 8s. (1·95 dollars) per ton. J. W.

THE ORE-DEPOSITS OF NAGYBÁNYA (HUNGARY).

Die Erzlagerstätten von Nagybánya in Ungarn. *By* GEYZA SZELLEMY. *Zeitschrift für praktische Geologie*, 1894, *pages* 265-271.

Rising suddenly out of the great Hungarian plain, the Wihorlet-Gutin chain, the "Erstwhile Burning Mountains," ranges from south-east to north-west at heights of 5,000 feet, and walls off the county of Marmarosch from those of Szatmar and Ugocsa, as well as from Transylvania and Bukovina. Here in Tertiary times took place the most violent and the most extensive volcanic outbursts that have ever been traced in the European area; and with the trachytic and tufaceous rocks, the long-cooled products of those outbursts, are associated many important ore-deposits. Messrs. Fichtel, Beudant, F. von Richthofen, Szabó, and Alex. Gesell have successively classified and described the rocks. These are chiefly amphibole-, pyroxene-, and biotite-trachytes, rhyolites, basalts, andesites, and propylites, the last-named being termed "greenstones" by the miners.

Along the whole range, wherever the andesite broke through the Tertiary and crystalline rocks, it dislocated and rent them, and in some localities the ores have concentrated in the fissures of the rocks thus shattered. It is observable in these cases that all indications of metalliferous ores disappear as one examines the rocks farther away from the andesitic eruptions. Later, the andesite was in turn burst through by the trachytes and rhyolites, was fissured and metamorphosed into "greenstone," and the cracks were filled by sublimation with ores, forming the numerous lodes which we now see.

The history of the very ancient mining industry in the Nagybánya district is briefly sketched, and the author then proceeds to describe:—

(1) *The Kreuzberg Mining-field.*—This covers an area of about 370 acres. The lodes have no salband; they consist of impure quartz impregnated with native gold in an extremely fine state of division, and with auriferous and cupriferous pyrites. Moreover there are "nests" and clefts filled with silver ores. The average metallic content per ton of ore is 25 to 30 grammes of gold, and 100 grammes of silver (about 0·81 to 0·97 ounces troy of gold and 3·3 ounces troy of silver per ton). A detailed table is given of the gold and silver production at Kreuzberg from 1863 to 1892, both years inclusive. The present average annual production of ore is 8,000 tons. A crushing-mill with 75 stamp-heads and 11 sweep-tables is at work.

(2) *The Veresviz (Red Water) Deposits.*—The stream which runs through this valley is deeply tinged with iron oxide. The lodes all strike nearly parallel north and south through the "greenstone" mountains of the Veresviz and four tributary valleys. The gangue here is calcite as well as quartz; in the latter, the native gold occurs in strings and flakes, often in well-formed octahedra and in model twins. In the pyrites, the gold appears to be chemically combined, as it cannot be obtained by amalgamation. Besides occurring in brittle ore, ruby ore, and as a

sulphide, silver is here found native in various forms. Out of 53 claims, 37 belong to the Hungarian Treasury; mining is at present conducted only by means of adits, and no deep workings have so far been attempted. The ten principal lodes are described in detail, and besides these a great many others are being worked. The annual production ranges from 4,000 to 9,000 tons. Connected with the mines by a 3½ miles railway are three crushing-mills, with 78 stamp-heads and 16 sweep-tables. The mills are now being fitted up with steam machinery and electric transmission of power, so as to make them independent of fluctuations in the water-supply. O. S. E.

THE ORE DISTRICT OF CINQUE VALLI.

Geologische Verhältnisse des Erzreviers von Cinque valli und Umgebung. By JOSEF HABERFELNER. *Zeitschrift für praktische Geologie*, 1894, *pages* 134-142.

A minute description of the rocks in which the principal veins of ore occur in the Cinque Valli district is given. It is considered to be a sort of contact-stone, that is to say a rock produced by a melting of the slaty rocks by the granite and a mingling of the two.

The mineral matter was originally disseminated through the rocks. Afterwards it was dissolved out by water, and redeposited in fissures. The metals represented are lead, copper, arsenic, etc.

The whole area from Roncegno to Pergine, through Fersina to the lake of Palu on the one hand, and from Pergine to the Pinè Valley on the other, may be considered a great ore-district. It is a favourable district for mining in the future as it has been in the past. At least 30 veins are known. G. W. B.

THE ORE-DEPOSITS OF FINLAND, RUSSIA.

Om Finlands malmförekomster. By AXEL TIGERSTEDT. *Vetenskapliga Meddelanden af Geografiska Föreningen i Finland, Helsingfors*, 1893, *vol. i., pages* 79-93, *with figures in the text and plates V.-VIII.*

The deposits of greatest industrial importance are the bog-iron ores, the deposition of which is still going on during our own day. The yearly production of cast-iron from these ores has throughout the century averaged a value of £50,000. The magnetic iron ores occur in lenticular nests in the most ancient rocks of the country: they are generally of poor quality, and not very abundant. Only two mines in Finland are at present working these ores. North of Lake Ladoga are fine examples of iron ore of eruptive origin, *i.e*, ore which has welled from the earth in a molten condition: the same origin is attributed to the magnetic iron pyrites with cobalt and nickel, which occurs south of Lake Saima. The author explains these as the result of "metallic liquation from the surrounding basic rocks."

He regards the copper ores of Orijärvi and Pitkäranta as having been vaporized or sublimated from the interior of the earth. At both localities the ore is visibly stratified. The ore at Orijärvi has been largely exhausted; but the workings at Pitkäranta are in active progress, the present production (annual) averaging 200 tons of metallic copper, about 20 tons of zinc, and rather less than half a ton of silver. Much zinc-blende is found in both mines.

In the north-eastern part of the duchy are innumerable cupriferous quartz-veins, but although they form a network over a large area, they seem too poor to be of much industrial value. In these the ore probably came in from the sides, in aqueous solution.

There are extensive gold-washings in Finnish Lapland, along the bottoms of the river-valleys. From 1870 to 1889 inclusive, the total gold production amounted to 11,552 ounces troy, the average proportion of the precious metal being 0·704 ounce for every 35 cubic feet of "slicks" washed.

On the whole, Finland cannot be regarded as a country rich in metalliferous ores, although most of the chief varieties are recorded as occurring there.

The paper is accompanied by tables giving the statistics of production of the iron-ores and copper-ores for a long series of years. O. S. E.

THE NORTHERN PORTION OF THE SIERRA DE USPALLATA, ARGENTINE REPUBLIC.

Estudios mineros en la Provincia de Mendoza: la Parte Septentrional de la Sierra de Uspallata. By G. AVÉ-LALLEMANT. *Boletin de la Academia Nacional de Ciencias en Cordoba (R.A.), vol. xii., pages* 131-176, *and* 3 *plates (maps and sections).*

The Sierra de Uspallata runs east of the Rio de los Patos, in the province of Mendoza, parallel to the Cordillera del Tigre of the Andes. The topography, orography, and hydrography of the region are described at some length, and the author then proceeds to deal with its geology. The rock-formations which constitute the sierra are the Huronian, the Lower Silurian, the Upper Trias and Rhætic, the Tertiary sandstones, and the alluvia, interrupted by or alternating with six several eruptive formations.

The Huronian includes much-cleaved phyllites of a silky lusture, chlorite-schists, a great spread of calcareous rocks (limestones, dolomites, and calc-schists), with dykes of diabase and great masses of serpentine. In the Quebrada de la Cortadera the fine colouration and veining of the serpentine are such as would make it eminently suitable for use in the arts. The rock is easily worked, and a market could be found in the country itself.

The Silurian, lying conformably upon the Huronian, consists of greenish grauwackes and slates, much folded, contorted, and dislocated, and dipping steeply to various points of the compass. The author found no fossils in these rocks. Upon them lie the variegated pebble-beds and sandstones (some plant-bearing, others gypsiferous), and the variegated marls (often bituminous and carbonaceous) of Upper Triassic or Rhætic age. Through these protrude dykes and sheets of olivine-diabase or palæodolerite. In the metalliferous mining district of Paramillo, coal occurs in appreciable quantity in these Rhætic strata, but not sufficiently to make it workable. In the Silurian grauwackes, but chiefly in the Southern Sierra, occur veins with auriferous galena of low grade.

Red sandstones and conglomerates, the latter containing huge boulders of trachyte, interbedded with flows of reddish sanidine-trachyte, unconformably overlap the Rhætic and Huronian. These strata are, presumably, of Kainozoic age.

A dark-red to purple orthoclase-porphyry attains a great development in the south of the area, and is associated with tuffs and breccias. The Tertiary eruptives comprise, besides the trachyte, tuffs mainly yellowish-grey in colour and hornblende-andesites. The latter are sometimes found injected into the dolerites (mentioned above), and they have metamorphosed the Rhætic beds even more completely than the dolerites have done. The masses of andesitic tuff intruded amongst the stratified rocks form the "country" of the metalliferous veins.

Auriferous quartz is known in two districts of the Northern Sierra, and has been worked for many generations, but the only mine open at the time of the

author's journey was that of Los Boques. There the chief veins cut through slate-coloured chert or lydianite, while at Agua de la Mina, in the Sierra Aspera, they cut through the andesite. The auriferous veins may therefore be classified in two of the categories of Prof. Groddeck, *i.e.*, the Australian-Californian type, cutting through metamorphosed sedimentaries, and the Nagyag type, running through igneous rock; yet both are here associated with the andesite formation.

Some of the abandoned gold-workings were to be reopened, as a company was engaged in setting up the necessary machinery.

Two deposits of copper-ore are known in the Northern Sierra. They both belong to the same type: thus at the Cañada Larga a mass of andesitic tuff is irregularly impregnated with copper silicate and malachite, but farther down the ore would appear to be copper pyrites. Native gold occurs of poor quality.

Silver-ores have been worked in the district since 1638, and the richest vein was the Vallejos; the workings in this have been long abandoned, but a company has now put down a shaft to cut the vein at a considerable depth. The other principal veins are the Alcaparrosa, Vetillas, and Trinidad. The silver-ores occur mostly in association with the dolerites, but some of the veins cut through the Rhætic plant-bearing sandstones and conglomerates. The system of veins spreads in the shape of a fan, the extreme radii of which are the Santa Rita Alta and the Alcaparrosa. The "fan" opens at an angle of 91 degs. 30 mins. to the west, and the maximum length of radius is 3½ miles. The veins are compound or multiple, consisting of several metalliferous leaders running, not always parallel to one another, through the encasing rock. Thus the total thickness of the Vallejos vein, even at a depth of 165 feet, attains 11½ feet; the Vetillas is 6½ to 10 feet thick; and the Alcaparrosa, 6½ feet. The smaller and very much narrower (7 to 19 inches) veins often do not exceed 300 feet in length, but they are none the less of industrial importance, because the ore in them is massive and abundant; these only cut through the dolerites. The vein-stuff is the same everywhere: massicot, spathose iron ore, hornstone, quartz, calcite, and seldom barytes. The ore itself is usually cerargyrite (silver chloride) and embolite (chlorobromide), while deeper down occur galena, blende, chalcopyrite, etc. Native silver is not of common occurrence. At the Carranza mine, a low-grade galena (0·55 per cent.) is worked, but the ore is abundant. The great enemy of the miner has been the water, which, accumulating in the bands of clay and tuff in between the dolerites, has broken into some mines over and over again; but new plant has been put down to overcome this difficulty.

The mining industry in this region is practically in the hands of the *Sociedad Minera del Paramillo de Uspallata*. They have now sunk four big shafts, and have erected works for treating the ores. O. S. E.

THE MINERALS OF BRAZIL.

By J. Ross. *The Engineering and Mining Journal* (*New York*), 1895, *vol. lix.*, *page* 125

Brazil has a great diversity of soil, minerals, and products; but, as other conditions are wanting, many of the minerals cannot be worked profitably. For example, such minerals as lead, copper, and iron ores, which occur, cannot be smelted for want of coal. Silver does not occur to a sufficient extent to pay for working.

A mine of quicksilver has recently been discovered in western Bahia, which may prove of considerable importance. Agates abound in the State of Rio Grande do Sul; some are exported to Europe, *vid* Montevideo, but there seems little demand. A pile of about 50 tons of agates has lain at the port of Salto Urugua for 10 years.

Diamond mining in Brazil was formerly carried on to a large extent. Now, it has greatly diminished, labour being dearer in Brazil and the price of diamonds having fallen through larger stones being found in South Africa. The Brazilian diamonds are hard, but comprise fewer large stones than the South African. Of the six most famous diamonds found in the world, Brazil has furnished two, the "Southern Star" of 254 carats and the "Diamond of the Crown of Portugal" of 120 carats.

The country is rich in gold-mines, which might be worked profitably if foreigners could secure legitimate protection in their mining rights.

The Brazilian gold-mines have all been, and still are, worked in an antiquated fashion, a profitable percentage of gold being lost with the tailings. Formerly, in the days of the slave trade, labour being much cheaper, gold-mining was carried on much more profitably. J. W.

THE ORE-DEPOSITS OF PUÑITAQUI (CHILE).

Die Erzgänge zu Puñitaqui in Chile, mit besonderer Berücksichtigung der Zinnober-führenden Lagerstätten. By A. GÖTTING. *Zeitschrift für praktische Geologie,* 1894, *pages* 224-230, *with sketch-plan in the text.*

The quicksilver deposits of this district of the coastal Cordillera are known to have been first worked in 1785 by the Spaniards, and, with very defective appliances, the ore extracted and treated on the spot yielded 4 per cent. of metallic mercury. After a break of 40 years, work was begun once more at the end of the 'fifties and then again dropped.

In the native tongue "Puñitaqui" is said to mean "gold land." It is certain that both natives and Spaniards at one time worked the auriferous and cupriferous lodes, but the industry perished in the revolutionary wars, and the existence of gold in the area under review was only re-discovered in 1880.

The mountain *massif* consists of syenites and diorites, interrupted by a broad belt of intrusive diabases (uralite-porphyrite and diabase proper). The latter shoot through the rocks in dykes, or spread out above them in sheets, and alter them at the contact into hornblende-granitite. Numerous instances of passage from one variety of diabase into the other are seen, and various stages of decomposition of the rocks are particularly observable in the neighbourhood of the metalliferous lodes. In basin-shaped depressions at the summit of the ridges are deposits containing a great variety of rocks, evidently derived from the main Cordillera, and probably smoothed by glacial action. It is in the diabase zone that the quicksilver ores and the auriferous and cupriferous lodes occur. Few lodes, and these of small importance, continue into the syenites and diorites; some are observed at the contact between diabase and granitite. Practically, the only gold and copper ores now worked thus occur at the contact in the southern portion of the area, at the Manto mine. There the gangue is a ferruginous auriferous quartz, with cavities containing gold-impregnated brown iron ore. The lodes are not thick, they join together frequently, are very variable in strike, and "pitch" steeply. Specular iron ore and tinstone have been found, but have not hitherto been worked, in the district.

The fifty or so known metalliferous lodes are grouped by the author in two categories. The first is the most important. The veins strike generally north and south, spread out northwards, and converge southwards to the diabase-granitite contact. The second, of less importance, strikes generally south-south-west and north-north-east, faulting the north-and-south veins. The average

thickness varies from 3¼ to 6½ feet, but where two systems of veins join, it reaches 10 to 18 feet. The state of decomposition of the "country rock" is remarkable, and appears to be entirely unaffected by such factors as relative depth or percolation of pit waters; and the more decomposed the "country rock" is, the richer the lodes appear to become. Quartz is the typical vein-stuff and the usual associate of the cinnabar, together with iron and copper pyrites, specular iron ore, and mercurial fahlore. Nearer the surface, malachite and azurite also occur. The author then describes in some detail the mineralogical aspect of the deposits.

The average percentage of copper in the good lodes is 12 to 14 per cent.; with only 8 to 10 per cent., the lode is considered unworkable. Workings are closed or re-opened according to the fluctuations in the market price of copper. The gold content averages 20 to 30 grammes per ton (0·65 to 0·97 ounces troy per ton) of vein-stuff. The deeper down the workings are carried, the more the gold content diminishes, while the proportion of iron pyrites increases, and the same complaint is heard as regards the percentage of copper in the Manto and Cerro de Tamaya mines. With regard to the quicksilver deposits, the French company to whom the author reported decided not to re-open them, as, despite the encouraging conclusions drawn from what he saw of the old workings and the old waste-heaps, he admitted that deep workings would afford little prospect of permanent success. Nevertheless he thinks that the gold-bearing lodes would repay working, as the mass of material which can be worked up is large, and the local conditions are favourable. In this case, as in that of the copper ores and in that of the so far neglected tinstone, the chief requisites are adequate capital and modern methods of working. O. S. E.

MINING INDUSTRY IN RUSSIA.

Aperçu général sur l'Industrie minérale de la Russie. *By* A. DE KEPPEN. *Annales des Mines*, 1894, *series* 9, *vol* v., *pages* 180-273 *and* 279-368, *plates VII.-VIII.*

The first few pages are devoted to a brief historical summary of mining administration and legislation in Russia from the days of Peter the Great downwards. Turning then to the actual condition of things, the author premises that in Russia "mining industry" not only implies quarrying of rock, extraction of ore, etc., but also metallurgy and chemical treatment of all mineral products, the one exception being the manufacture of (refined) petroleum which comes within the sphere of the Ministry of Finance. Mining administration in the Empire is directed by the Department of Mines of the Ministry of Domains; but the Minister for War looks after the mining affairs of the country of the Don Cossacks, while the Ministry of the Imperial Court superintends those of the Altai and Nerchinsk districts of Siberia. Finland is administered separately and possesses a Mining Code of its own, enacted in 1883. For the Empire as a whole, there is no complete and systematic Mining Code, but in volume vii. of the General Code of Laws are set forth the various enactments on mining as modified up to 1893. As to education, besides the great school of the Institute of Mines at St. Petersburg, where the highest grade of instruction is attainable, there are the secondary mining colleges of Ekaterinburg (Ural), Barnaul (Altai), and Dombrowa (Poland), and in the Donetz coal-field the master-miners' school at Lisichansk and the Poliakoff school. The number of these is now being increased by the new primary and secondary technical schools recently started under the auspices of the Ministry of Public Education. Moreover, a new school of mines is to be established at Irkutsk, for the especial purpose of training subordinate officials for working gold-deposits in Siberia.

The great and steady development of mining industry in the empire since 1825 is shown in a series of statistical tables. In 1891, the total value of the principal minerals extracted and treated metallurgically or otherwise reached the sum of £12,626,500. The number of workpeople employed was 440,517, and in comparison with the mining industry of other countries, that of Russia would seem greatly overmanned. But the reason is not far to seek: the widespread use of charcoal fuel in Russian works necessitates a vast amount of preliminary handling, and consequently an increased number of workpeople as compared with countries where coal is the chief fuel.

Dealing *seriatim* with the mineral resources of the Empire and the methods of working, etc., the author first devotes attention to:—

1. *Gold.*—In the Orenburg government, the Southern Urals are characterized by small and irregular placers worked by small syndicates. The same remark applies to those of the province of Perm, the only large placers in European Russia being those of Bogoslovsk, in the extreme Northern Urals. The placers of Western Siberia (Altai, Tomsk, Achinsk, and Minusinsk) are, generally speaking, poor—some workings barely suffice to keep a dozen or a score of men busy, no horses being required in such cases for the crushing apparatus. So, too, the Yenisei gold-deposits have seen their best days, and the remnants of their former wealth are just adequate for the maintenance of a small local mining industry. On the other hand, the placers of Nerchinsk, the Amur basin, Yakutsk, and the Lena basin are extensive, continuous, and rich, some of them yielding as much as 28,000 to 56,000 ounces of the precious metal annually. Comparatively speaking, very little gold is got in the Russian empire from auriferous quartz-veins. What mining there is of that description has developed much more rapidly in the Urals than in Siberia. In 1891, Russian gold-production amounted to 1,380,640 ounces, of which only 84,160 ounces represent metal obtained direct from auriferous quartz-veins. Many complex circumstances besides seasonal variations affect the annual production; among them, changes in taxation, fluctuations in the rate of exchange of the paper rouble, and alterations in mining legislation, play no inconsiderable part. The gold obtained from all mines or placers in the empire has to be sent to one of the three State smelting-works, established at Ekaterinburg, Tomsk, and Irkutsk, the cost of cartage thither being borne by the producer. The latter also pays for the additional freight to the Imperial mint at St. Petersburg and for the cost of minting the bars into coin. These expenses are, of course, supplementary to the royalties and taxes (payable to Government) which are practically proportional to the richness of the claim worked. The State banks advance money to the mining syndicates on the security of the crude metal.

The author gives several statistical tables regarding gold-mining in the Russian Empire, one of which shows the actual amounts produced and quinquennial averages from 1816 to 1891 inclusive. He also gives a summary of the legislation, and one notes that, although foreigners are equally entitled with Russian subjects to work auriferous deposits, the right is denied to the unfortunate Hebrew.

2. *Platinum.*—This metal is found in the Urals, near the boundary of the province of Perm, in placers which are often auriferous and are extremely variable in richness. The platinum occurs usually in small grains, and osmium and iridium are often found in conjunction with it. These placers of the Urals are the only known sedimentary platiniferous deposits, and recently platinum has been found in a mother rock (or true matrix) of chrome iron and serpentine. The precious metal is sent in the crude state to St. Petersburg, whence most of it is exported (without being refined) to foreign countries. In the table of exports during the years 1884-

1891, Great Britain appears as much the largest consumer until 1891, when the export to this country is marked as *nil*, and Germany takes the whole. There is in Russia a tax of 3 per cent. on platinum mined on private lands and 4½ per cent. on claims granted by Government. The enormous fluctuation in prices affects the amount produced, high prices constituting, of course, an incentive to work the very poorest placers. Platinum was used in Russia for coinage into 3, 6, and 12-rouble pieces between 1828 and 1845; but that use of it was abandoned in the last-named year, and the production immediately dropped to a very low figure until 1859, since which year it has steadily increased to the present annual average of 150,000 ounces; the extraction whereof employs 6,120 workpeople.

3. *Silver.*—All the silver got in Russia is extracted from argentiferous galena; but it should be borne in mind that all native gold of Russian origin is more or less argentiferous. Since 1831, the total silver production in the Empire has been slowly but continuously decreasing, although in particular districts, such as the Altai and Nerchinsk, it has begun to increase again of late years. The taxes bear the same proportion to value as in the case of platinum. The silver production in 1891 was 485,000 ounces (13,723 kilogrammes).

4. *Copper.*—The production of this metal attained its maximum in Russia between the years 1845 and 1852. At that time France appears to have been Russia's largest customer. The falling off since 1852 was attributed, among other causes, to the liberation of the serfs and to heavy taxation, indirectly favouring imports of foreign copper. In 1884, however, the Imperial Government imposed a protective tariff, and since then the copper industry has gradually revived, the total production increasing from 3,592 metric tons (3,535 British) of metal in 1882 to 5,455 metric tons (5,367 British) in 1891, an increase of 52 per cent., with 106 mines at work.

The two most important centres of the industry are the Bogoslovsk works in the Urals, and the Kedabek works in the Caucasus. At the first-named there is the necessary plant for enriching the ore in the wet way, the copper is extracted from its solutions by electricity, and the matt is treated by the Bessemer process. At the Kedabek works, petroleum-residues are used for heating the furnaces. On the western slopes of the Urals the ores occur in masses and nests in the Permian formation, while on the eastern they occur as veins in other formations. They are all of poor quality, but those of the Caucasus are rich, and those of the Kirghiz steppes (where work is at a standstill for want of fuel) are still richer. Russia, in 1891, imported nearly as much copper as she produced, about half coming through England and most of the remainder through Germany.

5. *Lead.*—All the lead got in Russia is extracted from argentiferous galena, which is worked in the Caucasus, Siberia, and the Kirghiz steppes. The total production of the metal in 1891 was only 556 metric tons (547 British). Russia has therefore to depend upon foreign countries for her lead supply, and the imports in 1891 averaged 20,000 tons, rather more than 55 per cent. coming from England. Plumbiferous ores have been recently discovered in the Governments of Arkhangelsk and Ekaterinoslav, associated in the latter case with zinc ores.

6. *Zinc.*—At present the production of this metal is confined to the southern part of Poland, although zinc ores are well known in other districts of the Empire. The Polish ores are calamine, lying in the Muschelkalk dolomites near the town of Olkusz. Two sheet-zinc works and a zinc-white factory have been established in Poland. The total production, however (some 3,200 tons of metal, 800 tons of white annually), is insufficient for the needs of the Empire, and 5,145 metric tons (5,061 British) of zinc was imported in 1891, most of which came from England.

7. *Tin* is worked only at Pitkäranta, in Finland, and the production is insignificant—9 or 10 tons annually. The imports average 2,000 tons every year, 70 per cent. being British.

8. *Mercury.*—In 1879, a rich deposit of cinnabar was rediscovered in the Government of Ekaterinoslav, near Nikitovka, on the Azov railway. It occurs as an infilling of fissures in sandstone of Carboniferous age. Mining was started in 1885, and the plant and machinery are of more modern type, and produce better results in proportion than the older works of Idria and Almaden. In 1891, with 746 workpeople, the production was 324 metric tons (318 British) of metallic mercury, about seven-eighths of which quantity was exported to Germany.

9. *Manganese.*—The most important workings are those of Charapanak, in the Government of Kutais, Southern Caucasus. They were started by large capitalists in 1879, but are now in the hands of a number of small owners, who are in very few cases expert in mining methods. The deposits cover an area of 31,616 acres, and occur as a band from 3¼ to 6½ feet thick, comprising six to eleven seams of pure ore, each of which is ¾ inch to 5 inches in thickness; they contain 56 per cent. of metallic manganese. A branch railway takes the ores direct on to the Transcaucasian main line.

Near Nikopol, in the Government of Ekaterinoslav, a manganiferous band, maximum thickness 20 inches, runs through Eocene strata. Two mines were opened there in 1886, and the assays show that the ore contains 57 per cent. of metallic manganese.

The total production of manganese ores in Russia in 1891 was 113,081 metric tons (111,263 British), whereof 100,000 came from the Caucasus. A very small quantity of ore is treated in Russia, most of the raw stuff being exported to the United Kingdom, Holland, and Germany.

10. *Cobalt and Nickel.*—There is a cobalt-works at Dashkessan, in the Government of Elisabetpol (Caucasus); it was started in 1867. Nickel, too, occurs in the Caucasus, in Daghestan, and at several localities in the Urals; but the most important nickeliferous deposits are those of Revdinsk (Ural), discovered in 1850. At that locality is the only natural nickel-oxide in Europe that is almost entirely free from sulphur and arsenic. The average metallic content of the ore is 2 per cent. Reliable statistics have not been obtainable for the period previous to 1874; since that year the total production has been 59 metric tons.

11. *Iron.*—The richest deposits of magnetite in the Urals are those of Blagodat (52 to 58 per cent. of metallic iron), Vissokaia (63 to 69 per cent. of metallic iron, no flux necessary), and Magnitnaia (66 per cent. of metallic iron). Deposits of less importance occur on the eastern flank of the chain, while in the north of the Urals a beginning has been made with working a specular iron ore, for treating which the Kutinsk works were built in 1890. The largest deposits of good brown hæmatite, yielding 60 per cent. of metal, and especially suitable for the Bessemer process, occur in the Central and Southern Urals. The red hæmatites of the western slopes yield 64 per cent. of metal, and chrome-iron ores, containing 35 to 38 per cent. of chromic oxide and 40 to 50 per cent. of iron, are worked at several localities. Ores of inferior quality, and generally containing phosphorus, are worked in Central Russia, and in the Government of Olonetz and the Grand Duchy of Finland. In Southern and Western Poland clay-ironstones, with 27 to 35 per cent. of iron, and brown hæmatites, with 35 to 45 per cent., are very extensively worked. These ores also are mostly tainted with phosphorus. In Southern Russia (Donetz-Kherson-Ekaterinoslav) are enormous deposits of very rich and pure magnetites, red hæmatites, and specular iron ore (60 to 68 per cent. of metallic iron); workings in them have developed astonishingly within the last few years.

In 1891, there were in the Empire 682 mines at work and 202 bog-iron ore workings ("lakes"), producing during the year a total of 2,958,378 metric tons (2,926,904 British). The fuel used in the manufacture of cast-iron is mainly wood charcoal, but in Central Russia and Finland firewood also is used, and peat in Poland. In the whole of Russia in 1891, only 16 blast-furnaces fired with mineral coal and coke were in blast (5 in Poland and 11 in Southern Russia), but the statistics show that these fuel proportions are in rapid process of reversal. The number of old-fashioned cold blast-furnaces fired with wood charcoal is yearly diminishing (50 per cent. of cold blast in 1882 and 31 per cent. in 1891), while modern plant and modern methods are asserting their mastery.

The iron production of furnaces fired with mineral coal, which in 1882 was only 7·7 per cent. of the Russian total, in 1891 was 32 per cent. Tables are given showing the progress of the Russian iron and steel industry from 1822 down to the present day, and they demonstrate, among other facts, that the production of cast-iron has more than doubled within the last ten years, while that of bar and sheet iron and steel has nearly doubled. All this increase must be credited to Southern Russia.

A succinct account is given of the various attempts to create and foster the steel-rail industry in the Empire. These were only attended with success after the Imperial Government's inauguration in 1876 of a system of bounties on Russian rails and duties on foreign rails

The Volga and its affluents, which, for the ironworks in the Urals, constitute the chief transport-routes, are closed by ice for four or five months of the year; while the ironworks of Poland, Central and Southern Russia, being in more direct communication with the great network of railways, are never thus cut off from their markets. Much of the iron made in the Urals is sold at the great fair of Nijni Novgorod. It is curious to note that Great Britain at the end of the eighteenth century was a relatively large importer of Russian iron. The imports into Russia of foreign iron and steel have been in process of diminution since 1878, partly because of the enormously increased customs duties, and partly because of the development by the Russians of their own iron and steel manufactures.

12. *Coal.*—The real rise of the coal-mining industry in Russia dates from 1855; up till that year Poland and Southern Russia produced between them roughly 150,000 tons of coal annually. Since then, not to speak of the Donetz coal-field, collieries have been opened up in Central Russia, in the Urals, in the Caucasus, and in far Eastern Siberia (mainland and Saghalien). On the railways alone the annual consumption of coal has increased from 498,358 metric tons (490,348 British tons) in 1875 to 1,388,086 metric tons (1,365,777 British tons) in 1891. On the other hand, the quantity of foreign coal so consumed has diminished from 286,868 metric tons (282,257 British tons) in 1880 to 102,521 metric tons (100,873 British tons) in 1891. Exact statistics as to the coal consumption in other branches of industry are not forthcoming. The total imports of coal and coke had not decreased much in 1891 in comparison with 1876, the annual average being steadily maintained at about 1,700,000 tons. The total production of coal in the Russian Empire in 1891 was 6,233,020 metric tons (6,132,846 British tons) from 323 collieries (950 shafts), employing 462 steam-engines and 40,225 workpeople (of whom 30,722 were at work below ground). The averages of production per pit and the average number of engines per colliery are much higher in Poland than in the Donetz and other coal-fields (the relation is about 10 : 1 for coal, 9 : 1 for engines). This enormous disproportion is partly explained by the fact that there is in Poland a settled population of experienced miners, and that most of the coal there is got from a

single seam varying between 25 and 50 feet in thickness, while in the Donetz basin the seams are rarely more than 6 or 7 feet thick.

The Donetz basin may be divided into three zones—the central and most extensive, where Carboniferous rocks come to day, and the eastern and western zones covered by more recent deposits, with here and there island-like inliers of Carboniferous. The total coal-bearing area of the Donetz is estimated at 4,942,000 acres, while that of the whole of Southern Russia is about double this figure. The country is gently undulating, cut here and there by deep ravines, and lies at an average height of about 500 feet above sea-level. As above-mentioned, the coal-seams are very rarely more than 6 or 7 feet thick, and the seams which have a thickness not exceeding 3 feet are, as a rule, quite pure coal, while the thicker seams are more or less intermixed with shale and sandstone. Roof and floor are mostly shales and sandstones. Almost every variety of coal occurs in this basin—gas coal, slightly bituminous and highly bituminous coals, semi-anthracites and anthracites. Statistics are given showing the enormous development of coal-mining in the Donetz since the days of the Crimean war. In 1891, there were 152 mines working anthracite and 125 working other coals, the former producing roughly 650,000 tons per annum, and the latter roughly 2,350,000 tons. The Donetz coal is not so much consumed within the district itself as exported to other parts of the Empire, the city of Odessa, which formerly depended on British coal alone, being a large customer.

The Central or Moscow coal-field covers an area of about 5,928,000 acres, and from it stretches a long strip of Carboniferous rocks through the governments of Olonetz and Arkhangelsk to the shores of the White Sea. The strata are predominantly limestones, the Coal-measures being intercalated between the lower limestones and the Devonian. There are only 12 outcrops now worked in the basin out of 200 known. The coal is difficult to work, often of bad quality, contains much ash, yields no coke, and cannot with advantage be carried to great distances or stored for any length of time.

The Kiev-Elisabetgrad basin is notable for its thick beds of Tertiary lignite, which extend into the province of Kherson, the total area of possible lignite-bearing beds being estimated at 1,235,000 acres.

The Polish coal-field, situated in the south-western part of the old kingdom of Poland, is in reality a prolongation of the Silesian coal-basin. The estimated area of coal-bearing strata in this field is 1,432,600 acres, but only about one-seventh of that area is in Russian territory, the remainder lying across the Prussian and Austrian frontiers. The surface of the country is somewhat hilly, with a general dip to the south-west, in which direction the older deposits are seen to come on from under the newer. The Carboniferous outcrops are surrounded by Triassic and Jurassic rocks, and occur at an average height of 900 feet above sea-level. Three great groups of seams are recognized, the chief of which, or Reden group, is the same as the Sattelflötz group of Prussian Silesia. The Reden seam itself varies in thickness from 25 up to 65 feet. As one goes westward, shales come in which split this great seam into three or four, the thinnest of them averaging 6½ feet; and this westerly variation is accompanied by a change in the character of the coal. In the Polish area the Reden seam yields no coal which is good either for coke or for gas manufacture. Twelve seams occur above it and nine below, the total thickness of the twenty-one seams averaging 108 feet.

The general strike is north-west and south-east, dip south-west, and outcrops are visible over a great extent of country. Fire-damp is, so far, unknown in the district, wherefore gunpowder is freely used for shot-firing. The strata

covering the Coal-measures are often very wet, and consequently inrushes of water into the workings constitute one of the chief dangers to be guarded against. Lignite-deposits occur in the Keuper (Trias) of Bendzin.

From 70,000 tons or thereabouts in 1855, the annual coal and lignite-production in Poland has risen to roughly 2½ million tons in 1891. Good hauling and pumping-engines and picking-screens are in use at all the collieries, and coal-washing apparatus at most of them. The larger number of pits communicate by branch railways with the great main line from Warsaw to Vienna.

On both the eastern and western slopes of the Urals, Coal-measures occur. Those on the European side resemble, by their intercalation between the Lower Carboniferous and the Devonian Limestones, the Coal-measures of the Moscow basin, but there the resemblance ceases. In the Urals, the coals are true coals, and not lignites and boghead as in the Moscow seams. On the Asiatic side are coals of various quality, also anthracites, graphitic coals, and graphites; and it was on these eastern slopes that the Ural coal-mining industry was at one time concentrated. It has now, however, migrated from the eastern to the western slopes. The average annual production is roughly a quarter of a million tons, mostly from four mines. The annual production of coke does not exceed 9,000 tons.

The Caucasus coals are of little industrial importance so far. In 1891, one mine was being worked in the Tkvibul deposits (30 miles from Kutáis) producing barely 7,000 tons. In the Northern Caucasus the Jurassic coals and bituminous shales of the Kuban district have been worked for well nigh fifty years.

The Kuznetzk basin in the south-east portion of the government of Tomsk (Western Siberia) is limited by the Saláir and Alatáu mountain-ranges, and cut in two by the river Tom. The area of the coal-field is close on 11,000,000 acres, and at many points thick seams of excellent quality have been proved, varying from 3 to 40 feet. The coal-bearing strata are of Jurassic age. In 1891, only two mines were so far at work, producing annually about 18,000 tons; but the quantity of available coal is estimated in millions of tons. Most of the coal got at present is used for the treatment of metalliferous ores in the Altái district.

In the Kirghiz Steppes are Carboniferous strata, with coals of various quality, lignites, and bituminous shales. Very little has been done to develop the mining industry there so far. Coal also has been proved at many localities in Eastern Siberia and Kamchatka. Especially noteworthy are the series of coal-deposits of the island of Saghalien, extending along the western coast for 590 miles, as well as along the eastern coast and inland. These coals are of Tertiary age, and nevertheless are said to vie in quality with the best Welsh coal. The seams rarely exceed 3 feet or so in thickness, and are worked by adits cut within the seam. The Saghalien coal is supplied to the Russian fleet in the Pacific, to foreign men-of-war and merchantmen, and the production is increasing to keep pace with the demand. One disadvantage is the distance of the coal-workings from the harbour of Vladivostok, and since 1886 the Imperial Government have organized a systematic search for coal on the mainland (maritime or seaboard province of Eastern Siberia). The best results are reported from the Suchan river, where is a seam, from 3 to 6½ feet thick, with coal of good quality, resembling in some respects that of Cardiff.

In Turkestan, coal was already known before the Russian annexation. Since then, further search has been made, with the view of supplying the flotilla on the Sea of Aral, and deposits are now being worked around Chimkend, Tashkend, and Khodjend. Some of the seams are 16 feet thick. In 1891, there were six mines at work, producing annually about 8,000 tons.

13. *Salt.*—Some 43 miles south of Orenburg is one of the most extensive rock-salt deposits in the world, that of Iletzkaia-Zashtita: there the rock-salt, very free from impurities, forms a mass at least 2 miles long and 425 feet thick, the estimated amount of possibly available salt exceeding 250 millions of tons. It belongs to the State, but is leased for mining purposes to private individuals. At Chapchachi, in the government of Astrakhan, is an equally enormous mass of rock-salt. In 1876, the Bakhmut-Brianzevka deposit in the government of Ekaterinoslav was proved: borings showed a thickness of 340 feet of pure salt. Rock-salt deposits are also worked at various localities in Transcaucasia and in the Transcaspian province.

A large number of brine-springs are pumped and the brine evaporated in various parts of Russia-in-Europe and in Eastern Siberia.

Salt lakes are very numerous in the Empire, and are scattered over a vast area. Among these may be mentioned Lake Elton (49,400 acres in extent), 185 miles from Saratov, on the left bank of the Volga, and Lake Bashkunchak (27,570 acres), besides seven hundred smaller lakes in the government of Astrakhan, dotted over the low-lying Arato-Caspian plain. In Lake Bashkunchak are three separate layers of salt, the uppermost of which alone represents a mass of more than 750,000,000 tons: a branch railway facilitates communications with the Volga.

In the Crimea, the salt-lakes are cut off from the sea by a very narrow strip of dunes, and the amount of salt harvested from them depends on the relative humidity of the seasons. Salt-lakes also occur in the governments of Stavropol and Kherson, in the country of the Don Cossacks, and in various provinces of Siberia.

The total annual production of salt averages 1¼ million tons (in 1891), the number of workpeople employed being 19,500. The excise duty on salt was abolished in 1881, but there is still a Customs duty on foreign salt of about 38 shillings a ton, except in the harbours of the government of Arkhangelsk, where the duty is only one-half of that amount. Glauber's salts are got from some of the lakes of the Caucasus and of Western Siberia, in all about 8,000 tons annually.

14. *Petroleum.*—The countless deposits, rich in petroleum, that occur in the Russian Empire are at once divisible into two categories: the first includes those deposits which are dispersed like islets amid the sea of surrounding strata, and the second those which cover vast continuous areas. Under the first heading come the Ushta River (Arkhangelsk) deposits, the Soké River (Volga) deposits, those of Kielce (Poland), of the provinces of Uralsk and the Turgái, of Turkestan and Saghalien Island. In none of these has much exploring work been done: indeed, they are so far all but unworked.

To the second category belongs the great petroleum-wealth of Russia, concentrated chiefly along the northern and southern slopes of the Caucasus, and in the governments of Elisabetpol, Tiflis, and Kutáis, the richest area of all being the peninsula of Apsheron, which runs out far into the Caspian Sea. East of that sea, moreover, in the Transcaspian province, rich oil-wells have been tapped. The author sketches the history of the development of the petroleum industry at Baku, etc., showing that its marvellous extension is of quite recent date, the deadening system of monopolies having been abolished only in 1872. The excise duties on the oils (higher per cwt. for light than for heavy oils) do not appear to have any deterrent effect, and they bring into the coffers of the Imperial Treasury over £1,000,000 yearly.

The annual production of petroleum in the Russian Empire increased from about 820,000 tons in 1882 to 4,700,000 tons in 1891, that is, it was nearly sextupled in the course of a decade; and the annual output of petroleum-derivatives was nearly quadrupled.

An observation which appears to confirm the geologist's view of petroleum-deposits, and tends to show that their probable limit of exhaustion is not so very remote, is that the petroleum has year by year to be sought for deeper and deeper down, while the average output per borehole becomes smaller and smaller; the average depth of borehole, which in 1881 was 420 feet, rose to 748 feet in 1891. The quantity produced by spouting (or artesian) wells has diminished in recent years by about 50 per cent.

Within the Russian Empire, the use of petroleum and its residues for firing the boilers of railway-engines, steamers, and factories, and even in the processes of iron-manufacture and copper-smelting, has greatly extended. Nor is it necessary to mention that imports of American petroleum are now reduced to vanishing-point. The financial result is that whereas Russia expended in 1873 about £1,050,000 on foreign petroleum, she received in 1891 more than £3,000,000 for her own petroleum: Turkey, Austria-Hungary, and Great Britain being her largest customers.

15. *Asphalte* is got from sandstones and limestones in the district of Syzran, government of Simbirsk, on the right bank of the Volga, and, under the name of *kir*, it is worked in the Caucasus.

16. *Sulphur.*—No detailed exploring work has been done in any of the Russian deposits. Near Sukéev, government of Kazan, on the Volga, sulphur is found in nests, and small agglomerations and bands in Permian limestone. The single refinery established there is now closed. Near Charkovo, government of Kiela, Poland, sulphur occurs abundantly, disseminated in Tertiary marls. There, too, a refinery has been established, and was at work in 1885, but has been closed within recent years. A new deposit has lately been discovered in the neighbourhood.

In the Caucasus, sulphur has been found at several localities in either side of the mountain range. Near Chirkat, province of Daghestan, a sulphur-deposit in the Upper Jurassic rocks is worked by a French company. Many other deposits are reported in the same province, and in the governments of Tiflis and Erivan.

Within recent years the discovery has been made of a very rich sulphur-deposit in the Transcaspian province, about 150 miles from Geok Tepé and Khiva; the amount of available sulphur is believed not to fall short of half a million tons.

So far, however, the total output of sulphur in the Russian Empire is exceedingly small, and large quantities of raw sulphur are imported from Italy, and of refined sulphur from France.

17. *Graphite.*—The famous deposits in the government of Irkutsk are now worked on a very limited scale, just sufficient to provide the Irkutsk gold-foundry with crucibles. The deposits discovered in 1860 in the north of the government of Yeniseisk, along the rivers Nijnaya-Tunguska and Kureika, are of excellent quality, and are being worked.

18.—The phosphate-deposits, building-stones, precious minerals, and thermal springs of the Empire are all briefly noticed; but the author has little that is new or of general interest to say about them. The appendix to the paper contains preliminary and incomplete mineral statistics for the Russian Empire in 1892.

O. S. E.

PREHISTORIC MINING IN THE ASTURIAS, SPAIN.

Las antiguas minas de cobre y cobalto del Áramo. By ALFONSO DORY. *Revista Minera*, 1893, *vol. xliv., pages* 333-7 *and* 361-6, *and* 3 *plates.*

The Sierra del Aramo, on the eastern slope of which the entrances to the prehistoric workings are found, is a spur from the main Cantabrian chain, attaining an elevation of over 5,500 feet above sea-level. It is a mass of Carboniferous Lime-

stone separating the basins of Mieres to the east and Quirós to the west. The workings are situated about half-way up the mountain.

When discovered in September, 1888, by Mr. van Straalen, the entrances consisted of narrow vertical chimneys, intended to prevent the escape of the slaves working within. In laying bare the rock for the foundation of the buildings for the present working, beneath successive layers of earth and clay numerous tools and other traces of human occupation were discovered. Moreover, near the village of Llamo, about ½ mile distant, considerable quantities of very homogeneous slag are met with, indicating that some sort of continuous furnace must have been used. The use of many of these tools is doubtful; some, however, were possibly used to crush and pulverize the ore. Small crucibles of clay, mixed with powdered quartz, have been met with. These sometimes contain fragments of partly-reduced ore, and are probably of earlier date than the furnaces above referred to.

Among the objects discovered in the workings may be mentioned sixteen skeletons, mostly of good stature, two of which are complete; stone hammers; horn picks; stone wedges; torches; two wooden bateas; a knife of bone (apparently human); a carved nut, etc. The skeletons, which are stained green with copper, appear to be those of young men. Examination, both of the crania and the method of working, suggests the succession of two distinct races. Some of the horns found refer to species which are extinct, entirely or in this region.

At the junctions of certain galleries a cross is traced, together with two horizontal parallel lines; these are referred to the second race of workers, and were possibly used as guide-marks. In another place a mass of white limestone is erected on a rough pedestal, so as to be visible from some distance, doubtless for the same purpose. There are traces of fire-setting having been used on a large scale to loosen the rock.

The clay which fills the cavities of the rock was scraped out by the workers' hands—this being shown by its containing thousands of impressions of all sizes, from that of a small boy's fingers to double the size of an ordinary man's thumb. In some places pillars are left, while in others masses of rock are piled up to support the roof.

Some of the galleries, driven in compact limestone, are so narrow that a man of average height can hardly squeeze through them, a fact which leads the writer to conjecture that the workers must have been excessively thin. In places the rock is polished by the circulation of the workers. In the inclined winzes there is no trace of steps or ladders; there is sometimes, however, a hole pierced in the rock near the top, from which a leather rope may have been suspended. The wooden *bateas*, one of which has a leather thong as handle, were doubtless used to convey the ore.

The width of the veins so far discovered varies from 1 to 6½ feet, the ores present being copper and cobalt. The former exists as black oxide, of from 15 to 25 per cent., coated with a skin of blue or green carbonate. The cobalt—also oxide—is black, hard, and compact, somewhat similar in appearance to the copper. It is disseminated through the dolomite walls, and especially near the cavities. The great regularity in direction and dip of the veins leads the writer to suggest that they were formed by the transformation of the limestone into dolomite bands by magnesian solutions, followed by others which deposited copper and cobalt in the cracks in the latter.

Particulars are added of the mineralogy and metallurgy of cobalt, especially the Herrenschmidt and Schœneis processes, and the paper concludes with an estimate of the commercial value of the deposits, which is considered to be very great.

G. E. C.

EXTRACTION OF ORE FROM WIDE VEINS OR MASSES.

By G. D. DELPRAT. *Transactions of the American Institute of Mining Engineers, 1892, vol. xxi., pages* 89-101.

The cross-cut system of mining is described, as carried out at the Cabezas del Pasto copper mine in the south of Spain, as opposed to the more common method of pillar-and-stall-workings, over which the writer claims several advantages.

The upper portion of the lode consists of gossan for a depth of 100 feet, composed of a siliceous peroxide of iron mixed with clay. The copper has been dissolved out and filtered into the undecomposed ore below, which is found to contain less copper with depth.

In the pillar-and-stall method, the lode is divided into floors about 35 feet apart, of which 12 to 15 feet has to be left as roof, besides pillars 15 to 20 feet square, so that only about one-half the ore can be extracted, while in the cross-cut system all the ore is extracted. At first two shafts are sunk, one for raising the ore, and the other for lowering the stowing material. The lode is 500 feet long, with an average width of 32 feet, and is divided into floors 65 feet apart, each floor being connected to the shaft by a cross-cut. When these cross-cuts have reached the lode, narrow galleries are driven, following the hanging-wall, to prove the shape of the mass. After this straight side-ties are driven for the more convenient tramming of ore to the cross-cuts and thence to the shaft.

Roads 6 feet square are now driven right through the lode to the foot-wall, at distances of 33 feet from each other. These are packed with stone, and other similar ones driven by the side of them or practically widened off the original road. This goes on until the stalls meet with the adjoining narrow road originally driven 33 feet away; thus the whole width of the lode is removed for a height of 6 feet.

The narrow gallery previously mentioned is then packed and another one started right above it, still following the wall of the lode, after which roads are again driven to the foot-wall exactly above the first ones, the only difference being that in driving the first roads the ore was solid, while in the second ones it is undercut and rests on the packing of the first.

The price paid per ton of ore for the first slice averages 2s. 1d., and in the first road, before the widening off had commenced, 3s. 1d., as opposed to 1s. and 1s. 9d. respectively when the lode had been undercut.

Slice succeeds slice until the floor above is reached, and here the packing has become so tight that no difficulty is met with. After the first slice has been removed the ore is thrown down chutes—carried up in the packing—into the cross-cut below, and thence trammed to the shaft.

The material used for stowing is quartzite and is quarried on the surface and lowered in hutches down a special shaft by means of double drums fitted with brakes: the loaded tubs pulling the empty ones up.

Each level is supplied with stone from the floor above by means of winzes made in the hanging-wall. Only large stone is used, as the small is much more expensive to handle down the mine and settles down so much when packed as to loosen the ore above and cause a treacherous roof.

The average quantity of ore broken by a miner is 3 tons per shift of 10 hours, as opposed to ¾ ton in the pillar-and-stall method. The average cost per ton, apart from general expenses, is 2s. 11d., and for stowing 1s. 8d. per cubic yard, which is equal to about 2½ tons of ore extracted. The cost per ton in the pillar-and-stall method averages about 4s. A. W. G.

THE SYSTEM OF FILLING AT THE MINES OF THE MINNESOTA IRON COMPANY, SOUDAN, U.S.A.

By D. H. BACON. *American Institute of Mining Engineers*, 1892, *vol. xxi., pages* 299 *to* 304.

The ore deposits occur in lenses, 200 to 1,000 feet long by 5 to 80 feet wide, lying at an angle of about 70 degs., with a vertical height of 250 to 500 feet.

The method consists in sinking a shaft in the country-rock with levels every 75 feet. Cross-cuts are carried in from the shaft, and the ore is worked each way from the shaft from foot to hanging-wall, and 15 to 20 feet high.

When this has been done, drift-sets consisting of cap and legs are set up the whole length of the opening, and connected with the cross-cuts. The necessary openings for ladder-ways and chutes are timbered, 5 feet square, from the floor to a few feet above the top of the drift-sets. Loose rock is then run in, until about 4 feet of loose rock is above the timber; then 10 feet of the roof is blasted down and more filling is let in, the ladder-ways, etc. being cribbed up. This is continued till the level above is reached. Raises, for filling, are driven in the foot and hanging-wall close to the ore, from level to level. In winter, when the rock at the surface is frozen, filling is obtained from the first level, or blasted down from the walls.

Several advantages are claimed over the system described by Mr. G. D. Delprat,* the chief of which are, that the expense of lowering the rock is avoided, and fewer rock-drifts or cross-cuts are required. A. W. G.

NOTES ON THE LEASE- OR TRIBUTE-SYSTEM OF MINING, AS PRACTISED IN COLORADO, U.S.A.

By B. B. LAWRENCE. *American Institute of Mining Engineers*, 1892, *vol. xxi., pages* 911-919.

The lease-system is becoming much more general as its advantages become better known.

The working of mines by corporations has often been attended with disaster, owing to the difficulty of securing a fair day's work from the men, the loss entailed by careless mining and sorting of high-grade ores, or by inefficient management.

All of these disadvantages can be in a great measure overcome by giving the men a stake in the mine, which is the main point in this system.

An owner through the causes mentioned above is unable to work his mine at a profit. He divides it up into a number of small leases, which are taken up by the men on the payment of a royalty upon the mill returns of the ore which they get. By dividing it up into blocks of ground, varying from 50 to 100 feet in length, and determined in height by the distance between the levels, each lessee is left to work his own particular block of ground, the ore being kept separate, and when delivered on the dump, becoming the property of the lessor or owner of the mine.

The ore is then shipped to the nearest sampling-works and its value ascertained, the lessee paying the cost of the control-sample.

The lessees are under the full control of the foremen, just as though they were working for day wages. The owner thus developes his mine at little or no cost to himself, the lessees driving levels, sinking winzes, etc., etc., in order that they may open up stoping ground. It is evident that under this method the owner takes little or no risk, and is assured of profit if ore be found. A. W. G.

* *Trans. Fed. Inst.*, vol. ix., page 513.

NITRATES IN EGYPT.

ANON. *The Engineering and Mining Journal (New York)*, 1895, *vol. lix., page* 81.

The report of the Commission to the Egyptian Council of Ministers dealt with the nitrate-bearing clay in Upper Egypt, which contains nitrates equal on an average to 15 per cent. of sodium nitrate. The supply is stated to be practically inexhaustible. Situated 470 miles (760 kilometres) south of Cairo, on the eastern bank of the Nile, are low ranges of hills at the foot of the principal range of limestone cliffs. These foothills were examined for a length of 15 miles, and are formed of nitrate-bearing clays, with a thin cap or covering of limestone. The clay is used by cultivators, locally, as a fertilizer. The clay is supposed to occur over an area stretching from 24 to 27 degs. north latitude, and from 28 to 34 degs. east longitude. Without the use of this *marog*, agriculture would be at a standstill between Assuan and Armant. The following analyses of *marog* were made by Dr. Mackenzie at the Tewfikieh College:—

Samples.	No. 1.	No. 2.	No. 3.	No. 4.
Moisture (dried at 212 degs. Fahr.)	2·44	1·83	3·42	2·68
Silica	19·34	10·37	22·56	22·81
Carbon dioxide	14·77	8·43	7·02	5·92
Phosphoric acid	0·57	1·72	0·29	0·27
Sulphuric acid	3·82	9·87	7·13	6·87
Nitric oxide	9·40	9·94	8·82	11·77
Chlorine	4·90	12·62	6·31	6·31
Iron oxide	4·37	4·37	5·65	5·27
Alumina	11·03	6·88	12·94	12·99
Lime	18·90	16·31	13·36	11·51
Magnesia	0·67	1·13	1·12	0·90
Potash	0·10	3·45	0·18	0·21
Soda and combined water (by difference)	10·74	15·77	12·55	13·84
Totals	101·05	102·69	101·35	101·35

J. W.

NYSTAGMUS, THE EYE-DISEASE OF COAL-MINERS.

Der Nystagmus der Bergleute. By DR. L. C. *Glückauf*, 1893, *page* 1391.

Nystagmus appears to have been first known about 1860, and it has been noticed that it has increased since then in all coal-mining countries. The principal symptom of the disease is that the miner, after a long shift, finds his sight unsteady, and his light appears to be in continual movement. A short interval of rest is sufficient to remove this unpleasant sensation. When the disease is more acute, the miner is attacked at his work, and finds that he cannot see to hold his tools; he has headaches; and, finally, cannot walk straight. Any attempt to see things by ordinary light above the horizontal level brings on the nystagmus. In one district, Dr. Nieden examined 11,145 men, and found that 3·6 per cent. suffered from nystagmus. The proportion between those coming from work and those going to work was as 4·1 to 3·1 per cent.

All those who have studied the subject agree that the disease is confined to those men who mine the coal, and who lie cramped up in a lying position, very often having to look upwards. The muscles of the eye cannot bear the excessive strain, which is aggravated by the poor quality of the light. It has been noticed that nystagmus is much more frequent in mines where safety-lamps are used than it is in mines where there are naked lights. As evidence of the important part that the quality of the light plays in connexion with the disease, it is said that at the Rhein Elbe collieries in 1877, when naked lights were used, 0·7 per cent. of the men suffered from nystagmus; three years later, after safety-lamps had been introduced, the percentage rose to 3·05.

The author has no doubt that the introduction of a better light in mines would reduce the disease. W. F. W.

A TEST FOR CYLINDER OILS.

ANON. *The Engineering and Mining Journal (New York)*, 1891, *vol. lii., page* 570.

Mr. T. H. Macdonald, chief engineer of the California Electric Light Company, San Francisco, has furnished some data respecting the quantity of oil used for internal lubrication of the engines at their Townsend Street Station. The areas given are square feet of surface moved over; that is, the internal surface of cylinders and valve-faces multiplied into velocity, as follows:—

I.—700 HORSE-POWER COMPOUND REYNOLDS-CORLISS ENGINE.
20 inches and 33 inches by 48 inches, 83 revolutions.

			Square Feet.
Valve-surfaces	multiplied into velocity	=	400
Surface high-pressure cylinder	,, ,, ,,	=	3,476
Surface low-pressure cylinder	,, ,, ,,	=	5,710
Total area oiled with 1 drop per minute			9,586

The engine was lubricated with one drop in two minutes with good results, showing a surface of 19,172 square feet by one drop of oil, but it is considered advisable to feed one drop per minute.

II.—600 HORSE-POWER COMPOUND SLIDE-VALVE ENGINE.
20 inches and 36 inches by 36 inches, 143 revolutions.

			Square Feet.
Valve-surfaces	multiplied into velocity	=	437
Surface high-pressure cylinder	,, ,, ,,	=	4,487
Surface low-pressure cylinder	,, ,, ,,	=	8,065
Total area oiled with 2 drops per minute			12,989

or 6,494 feet for one drop of oil. The feed on this engine was cut down to 4 drops every three minutes, or 9,742 square feet lubricated with 1 drop.

This indicates the marvellous viscosity of good oil, and, as a method of gauging the lubricating power of oils by the surfaces maintained, is new and commendable. Any engineer can count the drops of oil, calculate the area of the surfaces and multiply them into speed, and compare the results with the statements of agents for oils, which latter are often, of course, exaggerated. Apparently, there never has been, previously, a method capable of general use for arriving at any kind of measure of the efficiency of lubricating-oils. J. W.

PETROLEUM IN SOUTHERN STYRIA, AUSTRIA.

Notiz über ein Petroleum-Vorkommen in Südsteiermark. *By* DR. J. DREGER. *Verhandlungen der k. k. geologischen Reichsanstalt, Vienna*, 1893, *page* 287.

A small quarry, opened in the grey Upper Miocene sandstone of Friedau (Lower Styria), showed rock permeated and brownstained with bituminous matter. The burgomaster had reported the existence of a "petroleum spring," but the occurrence is, at any rate, quite local. The author found here fragmentary remains of a *Clupeus*, and he seems to infer that the petroleum originates from slow distillation of fossil marine organisms *in situ*. O. S. E.

PETROLEUM IN ALSACE, GERMANY.

Das Vorkommen von Erdöl im Elsass. *By* DR. JASPER. *Glückauf*, 1895, *vol. xxxi., pages* 215-218.

The author has had in his official capacity during the last 12 years ample opportunity for watching the development of the petroleum industry in Upper and Lower Alsace. Art. 1 of the Mining Law of December, 16th, 1873, enumerates bitumen (petroleum, mineral wax, etc.) among the minerals which are excluded from the landowners' proprietary rights, but for which mining concessions are granted by the State, and the necessary ground for working them may be compulsorily obtained. In practice, friction between the landowners and the mine-owners is rare, for the parties generally come to an amicable arrangement.

The first exploration for bitumen appears to have been made at Pechelbronn somewhere about 1745, and here workings for petroleum as well have continued down to our own days. An extraordinary outburst of activity took place in 1889, and borings were put down in every direction, through the Tertiary and Mesozoic rocks, to depths varying between 165 and 985 feet; they yielded a thick viscid crude oil of a brownish-black hue. The bituminous deposits, so far proved by 13 shafts and 700 deep borings, form a belt 25 miles long and 9 miles broad, ranging parallel with the Vosges; but there is reason to believe that the productive area will be found to occupy a much vaster extent of country. Especial reference is made in this connexion to the fish-bearing shales of the Altkirch district in Upper Alsace, which are distilled for oil, and correspond in geological position with the fish-bearing shales of Galicia; so that here, too, a natural reservoir of petroleum will probably be tapped below the shales. In Alsace generally, the bituminous bands are associated with the upper portion of the Lower Oligocene Marls. These are now known to be nearer 2,000 feet than 1,000 feet in thickness (as was formerly supposed). The attempt to work the bitumen and petroleum in underground workings by means of shafts was finally given up in 1888. Constant inflows of sand and brine, blowers of gases, accompanied by floods of petroleum oil, had invaded the workings at frequent intervals. Boring and pumping are the only methods now in use.

The physical and chemical properties of the Alsatian petroleum are as follows: – Its consistency (thickness) varies from place to place, and its colour from brown to black. The specific gravity is 0·82 to 0·95 at 32 degs. Fahr.; and boiling points range between 275 degs. and 302 degs. Fahr. Its percentage composition varies from 80 to 85 of carbon, 10 to 12 of hydrogen, 0·5 to 4 of oxygen, and 0·134 to 0·138 of sulphur. Moreover, it contains a good deal of paraffin, which separates out from it on reduction of temperature. The products derived from the crude oil may be classified as under:—2 to 3 per cent. of benzine and ligroine, 30 to 35 per cent. of

illuminating oil (petroleum, kerosene), 40 to 45 per cent. of residues (lubricating-oil, tar, gas-oil, paraffin-oil, vaseline), 10 to 15 per cent. of coke. The crude oil from the borings is pumped to the refineries of Pechelbronn and Sulz through a system of pipes laid in the earth over distances of respectively 3 and 4½ miles. In other localties similar pipes convey the oil to railway-stations, whence it is shipped in tank-waggons to various refineries in the Rhine Palatinate.

In pumping and boring all the newest appliances are used, preference being given to the Fauvelle system, the Canadian boring process, and the Raky system.

The annual production of petroleum in Alsace has steadily increased from 845 tons in 1878 to 15,632 tons in 1894. As before hinted, a much greater extension in the future may be looked for, and the author hopes to see Alsatian petroleum enter into active rivalry with the Russian and American oils for the supply of the inland trade in the German Empire. At present it only bulks for about 1·3 per cent. in this trade. O. S. E.

PETROLEUM WELL AT OBERKUTZENHAUSEN, ALSACE, GERMANY.

Note sur le Sondage d'Oberkutzenhausen, près Pechelbronn. By M. Mieg.
Bulletin de la Société Industrielle de Mulhouse, 1895, *pages* 107-114.

This boring, the work of the Société des Mines de Pechelbronn, is 1,670 feet (509 metres) in depth, and is lined with pipes decreasing in diameter from 15 inches at 54 feet below the surface to 5 inches at the depth of 1,621 feet (494 metres). Operations were performed solely with the freefall boring-apparatus. Down to a depth of 1,181 feet (360 metres) the boring passes through alternate layers of grey or greenish clay and siliceous sand, more or less impregnated with bitumen, and below these for a further 330 feet coloured and variegated clays, the detritus of Lias and Keuper rocks, are encountered. The petroleum stratum, a bed of sand about 3 feet in thickness, yielding, when pumped, 1½ tons of oil per diem, was struck at 1,097 feet (334½ metres), and at a further depth of 538 feet (164 metres) a brine-spring containing 8 per cent. of saline matter was tapped. This spring is artesian, and yielded at first 15,000 gallons of water daily, but the quantity has since decreased to 2,000 gallons. The water, like that of the Pechelbronn brine-springs, is rich in chloride (salt) and bromide of sodium, of which latter substance it contains up to 11·1 grains per gallon, and sulphates of lime and magnesia are also present. Curiously enough the temperature of the water is only 81 degs. Fahr., while that of the earth at 1,611 feet (491 metres) below the surface is 104 degs. Fahr.

Observations taken by Messrs. Schützenberger and De Chambrier show that the increase of temperature with the depth in this boring, although not absolutely regular, is abnormally high, amounting to 41 degs. Fahr. for 895 feet, which corresponds to 1 deg. Fahr. for every 22 feet, or three times the usual amount.

In considering the geological aspect of the region, and reviewing the theories propounded to account for the formation of petroleum, the author is of opinion that the fossil *Cyrenæ* and minute *Limnæ* found in the alluvial strata indicate that brackish-water or fluvio-lacustrine conditions prevailed at the time of deposition. The synclinal basin of this oil-field resulted from fractures sustained by the chain of the Vosges, and was gradually depressed during the Lower Oligocene period, the alluvial clays and sands from the denudation of the Trias and Lias being deposited in inverse order to the antiquity of the original formations.

While the scarcity of brackish or fresh-water fauna and the complete absence of marine fauna capable of producing the important accumulations of animal remains necessary, according to Prof. Engler's theory, for the production of petroleum, prevent us from affirming that the essential conditions on which the theory is founded are here met with, still the author considers that Prof. Engler's affords a better explanation of the formation of the Pechelbronn petroleum-beds than any of the other theories yet advanced. C. S.

PETROLEUM IN BURMA.

Note on the Chemical Qualities of Petroleum from Burma. *By* PROF. ENGLER. *Records of the Geological Survey of India*, 1894, *vol. xxvii., pages* 49-53.

The petroleum examined came from the following localities:—(1) Yenangyat; (2) Kodoung; (3) Twingon; and (4) Minbu.

It was examined chiefly with regard to specific gravities and products of fractional distillation.

The samples consist chiefly of hydrocarbons of the paraffin group (C_nH_{2n+2}), naphthenes occurring in small quantities only. The oil from Minbu is an exception, containing a large quantity of naphthenes. The composition of the different oils is as follows:—

	Light Oils (below 302° Fahr.).	Illuminating Oils (302° to 572° Fahr.).	Lubricating Oil and Paraffin Wax.	Difference and Loss.
Yenangyat ...	17·8 vol. %	... 49·4	... 30·8	... 2·0
Do. ...	18·6 ,,	... 49·6	... 28·3	... 3·5
Kodoung ...	4·3 ,,	... 38·8	... 50·4	... 6·5
Twingon ...	5·5 ,,	... 38·8	... 48·7	... 7·0
Minbu	—	... —	... 76·7	... 21·3

Hitherto the author has never met with oils of like consistency containing so high a percentage of illuminating-oil. G. W. B.

THE PETROLEUM INDUSTRY OF THE UNITED STATES.

L'Industrie du Pétrole aux Etats Unis d'Amérique. *By* Messrs. RICHE *and* ROUME. *Annales des Mines*, 1894, *series* 9, *vol. v., pages* 67-130, *and plate II.*

The authors were sent on a Government mission to the United States in the year 1892 by the French Minister of Commerce and Industry, and the data which they collected are presented in this paper with such clearness and conciseness that it may prove valuable even to the English reader. The first chapter contains a fairly complete summary of the chief occurrences of natural gas and petroleum in the States, together with statistics of production, number of wells, densities of oils, etc. The second chapter gives an account of the processes of refining the crude oil in use at the Bear Creek Company's factory near Pittsburgh, and at the Standard Oil Company's factories at Bayonne (New Jersey) and Whiting (Illinois), and the American processes are compared with the French, to the disadvantage of the latter. It is pointed out that the French refiners attempt by excessive heating to destroy the residual heavy oils and paraffin, so as to transform them into illuminating-oils; but these illuminating-oils are of so low a grade that they

deteriorate the good oils with which they are mixed for market purposes. The third chapter gives an account of the industrial applications of petroleum to purposes other than lighting or manufacture of lubricating-oils. A map of the petroleum and natural-gas region of western Pennsylvania accompanies the paper. O. S. E.

THE KEROSENE SHALE OF NEW SOUTH WALES.

Sur la Reinschia australis, algue permo-carbonifère qui a formé le Kerosene-Shale d'Australie. *By* C. Eg. Bertrand *and* B. Renault. *Association française pour l'Avancement des Sciences, Compte-rendu de la 22ème session (Besançon 1893), part 2, pages* 490-502.

The Australian kerosene shale or boghead is of Permo-Carboniferous age; it is very light (its specific gravity ranging as low as 1·05), very homogeneous, has a conchoidal, satiny fracture, is blacker than the Torbane Hill mineral, but of a duller black than Autun boghead. Flattened imprints of plants (*Glossopteris*, etc.) occur in it, and trunks of *Vertebraria* are recorded as being found erect in the shale at Joadja Creek. These tree-trunks are partially, and in some cases wholly, altered to jet: the authors think that the trunks are not *in situ*, but are driftwood impregnated with bituminous matter, such as the thelotite of Autun. On long exposure to the atmosphere and to rain, the more impure portions of large specimens of the shale assume a reddish tinge, and the laminæ of bedding then become visible.

The kerosene shale is the second known example of boghead almost entirely made up of an accumulation of gelatinous algæ in an ulmic precipitate. The structure of the alga is still recognizable, and the authors give to the species the name of *Reinschia australis*. It is calculated that a thickness of about 4 feet of the deposit represents 36,000 individual layers of algæ. The specimens examined enable the observer to trace the entire life-history of *R. australis*, through all its various stages of growth, from the young thallus to the adult; descriptions of the structures and measurements are given in full detail in the original paper. The transformation of the thalli into gum-like amorphous masses is noted, and attention is drawn to the fact that structureless yellow globules are the ultimate representatives of the thallus of a gelatinous alga. The remark is of importance in view of the study of the "yellow bodies" observed in coal. It is true that in the same shale empty and broken spore-cases give rise to another kind of "yellow body." The chemical composition of some layers of the kerosene shale differs from that of others; it is demonstrable that this difference is connected with the variations in size and condition of the thalli of *Reinschia* forming those layers. Where the boghead seems purest, there the proportion of alga (as compared with the intervening ulmic precipitate and vegetable detritus) is predominant.

The authors conclude that the kerosene shale is a carbonaceous formation of the same type as the Autun boghead. Aquatic plants covered the surface of peaty pools, and rained down on the bottom where was accumulating the ulmic precipitate from those waters. Around the pools grew a cryptogamic vegetation, sufficiently abundant to give rise to showers of spores. Brown matter, of bituminous nature, clearly distinct from the ulmic products, commingled with the mass shortly after, or perhaps during its deposition. The rate of accumulation of the deposit was rapid. Unlike the Autun boghead, it contains no fish-scales, nor have the authors found in it any coprolites or siliceous nodules. O. S. E.

PHOSPHATE-DEPOSITS.

Bibliographie générale des gisements de phosphates de chaux. *By* X. STAINIER. *Annales de la Société Geologique de Belgique*, 1893, *vol. xx., Bibliographie, pages* 3-28.

This paper contains a list of works relating to phosphatic deposits for the following countries:—

Europe: Austria, Belgium, England, France, Germany, Italy, Norway, Portugal, Russia, Spain, Sweden; Asia: India, Palestine; Africa: Algeria, Tunis; America: Argentine Republic, Canada, Peru, United States; The West Indies; Pacific Islands.

The particulars for each country are given separately and in alphabetical order according to the name of the author. G. W. B.

PHOSPHATE-DEPOSITS IN SOUTHERN FRANCE.

Notes sur la nature et le gisement du phosphate de chaux naturel dans les départements du Tarn-et-Garonne et du Tarn. *By* CHARLES HELSON. *Annales de la Société Géologique du Nord*, 1893, *vol. xxi., pages* 246-265.

The introductory portion of the paper is devoted to a general description of natural phosphates of lime from the geological, mineralogical, and chemical point of view. Phosphate derived from bones is also touched upon, and the use of phosphates in agriculture is then dealt with.

In Tarn-et-Garonne the deposits were first discovered in 1862 on the Jurassic limestone-plateau above Caylus by the late Mr. Poumarède; but, although samples showed on analysis an equivalent of 70·64 per cent. of pure tribasic phosphate, no attempt was made to start workings until the end of 1870. Besides those near Caylus, important deposits occur in the neighbourhood of Bruniquel, Caussade, St. Antonin, and Montcéré (Tarn). British and Belgian prospectors leased some of the claims at extravagant prices, which in many cases swallowed up all the profits. In fact the deposits are only workable at a profit by chemical manure manufacturers, who deliver the prepared article besides extracting the raw material. This is due, as the author believes, to two main causes:—(1) The heavy royalty payable to the owners of the land, which is in some cases as much as £1 per ton of phosphate extracted. In others the royalty is 17s., in yet others 10s. and 8s., say on an average 13s. per ton. This royalty, however, is not due unless the raw stuff contains at least 70 per cent. of tribasic phosphate. If the "ore" is too poor, the lessee may either leave it or pay to the owner a compensation tax of about 4s. or 5s. per ton. The ordinary duration of leases is 10 years, which term may be extended by agreement. (2) The mode of formation of the deposits, which all occur on the surface of limestone-plateaux of middle Oolitic age. The phosphate is non-crystalline, generally pale and whitish in colour, sometimes tinged with grey, red, and yellow; the structure is concretionary. The phosphate fills pockets, clefts, and pipes which have evidently been produced in the limestone since the consolidation of that rock. It seems probable that the phosphate was precipitated from mineral waters containing abundant carbonic acid, whose presence increased their corrosive action on the limestone.

The observations made around Bruniquel and Caylus show clearly the distinct facies to which the deposits belong. For example, near Caylus, the irregular cavities or pockets filled with phosphate, which are often only a few yards across,

attain 65, 100, and even 115 feet in diameter. The bones and pebbles found in the pocket-deposits were evidently brought in from no great distance in some easily imaginable way, and their presence does not at all prove that the water in the "basins" constituted a medium which could sustain life. They show, however, that the deposition took place in the Tertiary period, and continued for some time during that period. Elsewhere the deposits are elongated "veins" with approximatively parallel vertical walls, which generally suggest infilling of rectilinear crevices. But these again are to be distinguished from the fault-fissure-veins, because they narrow very rapidly downwards, and surface indications are therefore extremely deceptive as to the mass of a deposit. Some of the deposits strike east-and-west, and another series crosses them at right angles, striking north-and-south.

The number of workings at present is small, for one thing, because in the desire to avoid competition and to monopolize for themselves the benefit accruing from recent discoveries, the present workers in the field have not bruited much abroad the fame of the Tarn-et-Garonne phosphates; and for another, because land-owners, tempted by speculative bids, have become more and more exacting in the matter of royalties, and have thus discouraged intending lessees.

Ten detailed chemical analyses are given showing respectively the following equivalents of tribasic phosphate:—85·72, 80·99, 80·34, 78·36, 76·62, 71·55, 69·79, 65·17, 62·43, and 58·07 per cent.

Not only in the neighbouring district is the phosphate used for agricultural purposes, but it commands a sale throughout a large portion of France.

In the same volume of the *Annales* is a translation by Mr. V. Vaillant of Mr. G. H. Eldridge's paper on the phosphates of Florida. O. S. E.

PHOSPHATIC CHALK IN THE NORTH OF FRANCE.

1.—*Note sur les gîtes du Phosphate de Chaux de Templeux-Bellicourt et de Buire.* *By* J. GOSSELET.

2.—*Note sur les gîtes de Phosphate de Chaux aux environs de Fresnoy-le-Grand.* *By* J. GOSSELET. *Annales de la Société Géologique du Nord*, 1893, *vol. xxi., pages* 2-10, 149-159, *with sections in the text.*

In the autumn of 1892, Mr. Gosselet visited the phosphate-workings of the district of Templeux-Bellicourt, situated near the boundaries of the departments of the Somme and the Aisne. They extend over an area, a little more than 3 miles long and ½ mile wide. At Villeret quarry is a section showing the grey chalk-with-phosphate intercalated in the white chalk: and in the former the amount of phosphate increases downwards, with a stratum of phosphate-nodules towards the bottom. It is inferred that these nodules were formed contemporaneously with the chalk. The workings are mostly in the phosphatic sand, which represents the residue from the dissolution of the chalk. Sometimes this sand occurs in horizontal layers on the surface of that formation (as at Les Hautes Bruyères), but most frequently it fills "pockets" in it, exactly like those known near Doullens. The author describes and figures a section at Templeux showing that chalk-rubble was formed from the white chalk at some period between the deposition of the phosphatic sand and the overlying Tertiary beds; the latter have here a line of green-coated flints at their base.

In the Buire district, besides phosphatic grey chalk (6½ to 9½ feet thick), and phosphatic sand, the nodules become so abundant as to form a kind of conglomerate, with an analysis showing 81·18 per cent. of phosphate of lime. It is

suggested that this conglomerate may be the result of a pre-Tertiary resorting or re-arrangement of the phosphatic chalk-beds. The "pockets" in the neighbourhood of Buire are much shallower than those worked at Doullens.

Near Fresnoy-le-Grand, at Montbrehain, phosphatic sand occurs in "pockets," which in very few cases go further down than the grey chalk; they are usually from 3½ to 5 feet deep, and 15 inches in diameter. When they reach into the white chalk below, the surface of that rock is found to be coated with black clay.

At Fresnoy-le-Grand itself, the phosphatic grey chalk (5 to 6½ feet thick) is, like that at Villeret, observed to grow richer downward. Below it is a conglomerate (15½ inches thick) of phosphatic nodules in loose chalk, which passes into sand. Fossils are seen adhering to the surface of the nodules. The analyses appear to show that the nodules and the accompanying chalk-beds are very much poorer in phosphate of lime than the corresponding deposits at Buire.

At Méricourt, ½ mile south of Fresnoy quarry, the phosphatic chalk is again exposed, but here the conglomerate of nodules has in places thinned out, and in places completely disappeared.

Workings were just being begun at Étaves, and there the phosphatic sand-pockets in the grey chalk are overlain by Tertiary beds analogous to those seen at Templeux. The grey chalk is here 5½ feet thick, and is worked for phosphate as well as the sand.

[The detailed descriptions of all these occurrences, which are found in the original paper, together with the numerous diagrammatic sections in the text, will probably throw light on similar occurrences in other districts, and in view of the growing industrial importance of phosphate-deposits all over the world, no apology seems needed for devoting some space to the subject here.—EDITOR.]

O. S. E.

PHOSPHATIC NODULES IN THE LIAS OF LORRAINE, GERMANY.

Sur le Gisement et la Structure des Nodules phosphatés du Lias de Lorraine. By — BLEICHER. *Bulletin de la Société Géologique de France*, 1893, *series* 3, *vol. xx., pages* 237-247, *and plate VI.*

After a brief reference to previous work on this subject, the author gives a table of the succession of the Liassic strata in Lorraine.

In the Lower Lias two horizons of phosphatic nodules occur, with casts of fossils, in marly limestones and sandy marls, but would not repay working.

In the Middle Lias there are four horizons of phosphatic nodules, three of which are worked in the department of the Vosges.

In the Upper Lias are marls with phosphatic nodules, which would not repay working. The uppermost beds are sandy marls with iron ores.

Where, as in the Vosges, it pays to work these phosphatic nodules, they are found lying heaped together loosely, freed from their matrix. Thus the compact ochreous limestone of the Middle Lias, which in Meurthe-et-Moselle protects the subjacent marls with phosphatic nodules from denudation, and thereby deprives them of industrial value, gives place laterally, as one approaches the department of the Vosges, to a sandy, easily-crumbling marl; whence the nodules are washed free and the deposits are "naturally concentrated." Near Oëlleville and Totainville there is a band of nodules, 12 inches thick, below 8 feet of ferruginous sandy marl overlain by humus. These 9 feet of deposits represent the residues of the "natural elutriation" of about 110 feet of strata.

A somewhat detailed description is given of the microscopic structure of the nodules, some of which are largely made up of casts of bivalves and ammonites,

others of fossil spongida, foraminifera, shell-débris, etc.; and attention is drawn to the dissimilarity which they exhibit to the phosphatic nodules dredged from great depths in the ocean by Dr. John Murray. They contain no manganese, nor have they a continuous ferruginous external crust. The nodules in the Upper Lias only contain 13·5 per cent. of phosphate of lime, and are generally smaller and more irregular in structure than those of the Middle and Lower Lias. None of these Lorraine nodules bear any resemblance to coprolites. O. S. R.

PHOSPHATE-DEPOSITS IN ALGERIA.

1. *Description géologique de la région des phosphates du Dyr et du Kouif, près Tebessa.* *By* — BLAYAC. *Annales des Mines*, 1894, *series* 9, *vol. vi.*, *pages* 319-330, *and plate X V.*
2. *Note sur les lambeaux suessoniens à phosphate de chaux de Bordj Redir et du Djebel Mzeita.* *By* — BLAYAC. *Annales des Mines*, 1894, *series* 9, *vol. vi.*, *pages* 331-337.

North-east of Tebessa, a few miles from the Tunisian border-line, are two outliers of Suessonian (Lower Eocene) limestones, rising as table-shaped hills some 2,500 feet above the plain; these are Jebel Dyr and Jebel Kuif. There is a third outlier, Ain Kissa, let down by faults south of the first, and phosphate workings have just been started in it.

The author describes briefly the succession of Cretaceous and Eocene strata observed in these hills in ascending from the plain to the summit. The Lower Eocene consists of black and grey marls at the base, overlain by marly limestones with siliceous nodules and phosphate-bands, and these in their turn are capped by hard compact limestones, containing *Nummulites* and *Thersitea*. The Eocene here is lying in two synclinal folds of *Inoceramus* limestones.

The marls have at their base at Ain Kissa a band, 27 to 31 inches thick, of phosphate of iron, with selenite-crystals and sharks' teeth. This band appears to occur at no other locality, but the marls are slightly phosphatized throughout, and make excellent arable soil.

In the marly limestones at Jebel Dyr are three beds of phosphate of lime, the lowest being about 10 feet thick, the next 2 feet, and the highest 20 inches thick. The 10 feet bed is highly siliceous in its upper portion, and this hard, siliceous part is used as a roof for the workings. The average percentage of phosphate in the rock that is mined varies between 60 and 75. The two other beds are not worked; followed westward, they are seen to join, and then to thin out altogether.

At Jebel Kuif there is only one bed of phosphate between the black marls and the upper limestones, but it is 14¾ feet thick, and at least three-quarters of this thickness is workable. Some of the softer portions average as much as 78 per cent. of phosphate. At Ain Kissa there are as many as six separate bands of phosphate interstratified with the marly, siliceous-nodule-bearing limestone, but they cover a comparatively small area, being largely cut out by the numerous faults which have dislocated this outlier.

The second paper, by the same author, records the occurrence at Jebel Mzeita and Bordj Redir of strata similar in age and character to those above described. At the former locality, four bands of phosphate-rock occur in the marly limestones, but this rock is harder than at Jebel Dyr, black and speckled with white and grey (Coprolites). The thickest band is about 6½ feet, and analysis has shown the phosphate percentage to average 66·5. The Bordj Redir outlier may be aptly compared with that of Ain Kissa, exactly as Ain Kissa has been let down by faults below the Jebel Dyr, so has Bordj Redir been let down below Jebel Mzeita.

There are here as many as eleven distinct bands of phosphate, one of which is 6½ to 10 feet thick. At Bel Imur are phosphatized shaly marls in the long 25 miles outcrop of Lower Eocene, but as one travels northward these phosphatized marls are seen to pass gradually into marly limestones. However, along a line drawn from Sukahras to Bordj-bu-Arreridj, 220 to 250 miles, there is a chain of these Suessonian outliers, all containing phosphates. These deposits, similarly to those occurring between Tebessa and Batna, are near the shore-line of the Suessonian sea, and their fossil contents, as well as their other characters, show that they were laid down in comparatively shallow water. O. S. E.

THE PHOSPHATE-DEPOSITS OF ALGERIA.

ANON. *The Colliery Engineer (Pennsylvania)*, 1895, *vol. xv., page* 153.

Mr. Chas. T. Grellet, U.S. Consul at Algiers, reports to the State Department the working of important phosphate deposits in Algeria, which will, in the near future, seriously affect the European markets of American phosphates. Since 1886, the French explorers of Western Tunisia have mentioned the existence of large phosphate deposits in that country; and about two years ago similar discoveries were made in the department of Constantine (Eastern Algeria) in the region of Tebessa, about 155 miles south of Bone.

The mining of the phosphate is said to be exceeding easy and cheap, the Tebessa fertilizer being composed mostly of the bones and teeth of sharks, and being found in a semi-pulverulent state, in strata from 20 inches to 10 feet thick, intercalated in white marl and siliceous limestone.

Its strength varies from 60 to 70 per cent., and in some places reaches as high as 85 per cent. of tribasic phosphate of lime.

The phosphate-lands of Tebessa are now under the control of three firms, two of which are English, viz, Messrs. Crookstone Bros., who own Jebel Dyr, and the Constantine Phosphate Company, whose head office is at Leith, in Scotland.

The general output of these mines was about 5,000 tons in 1893; whilst this year (1895) it is expected to attain more than 30,000 tons. J. W.

THE PHOSPHATE-MINES OF CANADA.

By H. B. SMALL. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxi., pages* 774-782.

There are two main districts where apatite is mined: Quebec, the most important, and Ontario, which is not so extensively developed.

The rocks containing apatite are composed of pyroxene, quartzite, mica, gneiss, and crystalline limestone.

The apatite, or phosphate, as it is commonly called, occurs in a variety of forms, in crystals, or masses, in strata, and most commonly in a friable form known as "sugar-phosphate." When mixed with foreign substances it has to be cobbed by hand, which is rendered easy owing to the friable nature of the phosphate. The great drawback to this industry is the absence of sufficient quantities of pyrites in the neighbourhood for the manufacture there of sulphuric acid, which is used to convert the crude phosphate into a fertilizer.

The crude material is floated down the river Lievie, a tributary of the Ottawa, to Buckingham, where it is loaded into trucks and conveyed by rail to Montreal, whence it is shipped in great quantities to England and the European continent, very little of it finding its way to the United States.

That the industry is a growing one is shown by the increase in the exports from Canada, which in 1879 were 8,446 tons, and in 1889, 29,987 tons. A. W. G.

PHOSPHATE CHEMISTRY AS IT CONCERNS THE MINER.

By THOS. M. CHATARD. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxi.*, *pages* 160-175.

The writer describes the analysis of a natural phosphate, and urges the desirability of a standard method for analysing; pointing out the conflicting results arrived at by different chemists. A. W. G.

THE GEOLOGICAL ORIGIN OF PHOSPHATE OF LIME IN THE UNITED STATES AND CANADA.

By WALTER B. M. DAVIDSON. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxi.*, *pages* 139-157.

The apatites of Canada occur as pockets in bands of pyroxene, overlying limestones of Laurentian age. The phosphoric acid was derived from the limestone which was subjected to great heat and pressure—causing folding—and the apatite crystallized out in pockets or cavities found along the lines of least resistance during the period of contortion.

Phosphate of lime has a curious faculty of motion and a great tendency to aggregation. The phosphate-beds of Carolina were deposited by a river flowing through a district of Vicksburg Limestone, whence the calcareous phosphate has been derived. The prevalence of fossils is due to the preserving action of the phosphate of lime, and the phosphoric acid in the beds was not derived from the bones, as was generally supposed, but most of that in the bones was derived from the beds.

The State of Florida is underlain by Eocene or Vicksburg Limestone with later Tertiary clays and sands. The limestone has been largely denuded, and it is from this that the various phosphate-beds derive their origin.

Nearly all deposits of phosphate of lime belong to the Cretaceous or later periods, and are either phosphatic limestones or chalks, or else a deposit of phosphate of lime lying on a limestone or chalk.

The same action of the various forms of life that secreted the carbonate which makes up the bulk of a limestone or chalk was at work extracting the phosphate from the waters of the ocean.

The writer was very much struck with the similarity of the West Indian and Florida phosphates and thinks that the origin of both is the same. A. W. G.

FLORIDA PEBBLE AND NODULAR PHOSPHATE OF LIME.

By PROF. E. T. COX. *Engineering and Mining Journal* (*New York*), 1891, *vol. lii.*, *pages* 359-360.

The author read a paper, at the Indianapolis meeting of the American Association for the Advancement of Science, on Florida hard rock-phosphate, when he applied the name of floridite to this mineral. The name was given chiefly on account of the physical differences that exist between it and other phosphates of lime, and the source of its origin. The author then explained that, in his opinion, the hard rock-floridite had been derived from the mineralization of ancient guano.

Along with the beds of hard rock-floridite are pockets of soft phosphate of lime, which latter has the appearance of a soft plastic clay, its colour varying from greyish-white to a kaolin-white. When dry, the soft phosphate yields from 75 to 84 per cent. of $Ca_3(PO_4)_2$. The beds of hard rock-phosphate follow the trend of the Gulf

of Mexico on the western side of the peninsula of Florida. They are of the Quaternary age and rest upon Eocene beds, sometimes attaining a vertical thickness of 60 feet or more, but generally much less.

The beds of floridite are not in a solid continuous stratum. They cover an area from Taylor and Madison counties, in the north, southward to Citrus county, forming a strip about 120 miles long by about 20 miles wide. The land- and river-pebble and nodular phosphate-belt commences south of Citrus county, and is traced as far south as Caloosahatchee river in Lee county. The pebble-phosphate occupies the same geological position as the floridite; it, too, follows the trend of the Gulf of Mexico, and is in non-continuous beds, which extend over an area of about 36 miles. The stratum of pebble-phosphate ranges from 3 to 30 feet in thickness, 12 feet being the average thickness in most parts. The pebbles contain from 70 to 80 per cent. of $Ca_3(PO_4)_2$, and their clay matrix contains about 50 per cent. of the same constituent.

In and near the town of Bartow, Polk county, there are found, resting on the pebble-bed, bones of the manatee and other animals.

The pebble-bed is remarkably free from fossils, but the author has found a few small sharks' teeth in it. The pebbles vary in size from that of a mustard seed to that of a hen's egg. Their colour varies from black to brown and milk-white. The phosphate of iron and alumina in the pebbles is rarely found above 3 per cent.

Dr. Henry Wurtz analysed a specimen of the nodular phosphates from Polk county, in the Peace river region, with the following results:—Tricalcium phosphate, 72·79 per cent.; aluminium phosphate, 3·28; ferrous phosphate, 3·08; calcium fluoride, 8·90; calcium carbonate, 0·41; calcium sulphate, 2·32; combined water, 2·64; organic (bitumenoid) matter, 0·44; quartz, 5 55; soluble silica, 0·12; total, 99·53 per cent. The specific gravity of the whole was 2·765, and of the compacter fragments 2·828. J. W.

PLATINUM IN CANADA.

Canadian Platinum. *By* J. T. Donald. *The Engineering and Mining Journal* (*New York*), 1892, *vol. liii.*, *page* 327.

It has long been known that platinum had been found in this province. In the Report of the Geological Survey for 1851-52 it is stated that Dr. Hunt had detected native platinum in some of the gold-washings of the Chaudière district.

Canadian platinum-ore, as a commercial article, is entirely the product of British Columbia. In association with alluvial gold it has been met with in a number of the streams in that province.

At present, the most important platinum-bearing district of British Columbia, as well, indeed, as of North America, is that of the Tulameen or North Fork of the Similkameen river. A notable quantity of platinum has already been obtained from this district. One firm in the United States claims to have purchased within the last year or two fully 2,000 ounces of British Columbia platinum, and it is well known that a portion of the yield of this district has found its way to the London market.

An increased output may be expected, as the Tulameen Hydraulic and Improvement Company has made preparations to begin hydraulic mining on a large scale with the advent of spring. Two samples of the Tulameen ore contained 69·28 and 72 per cent. of platinum respectively. It is really a very complex alloy of platinum, with a number of the comparatively rare metals of this group, such as

palladium, iridium, and notably an alloy of osmium and iridium, known as osmiridium, which, in grains of proper size and form, is used for pen-points. This ore is worth to-day about 23s. (5½ dollars) per ounce troy.

The price is very unsteady, being determined by the demand for the metal and by the state of affairs in Russia, the principal producer of the ore. J. W.

A VARIABLE-SPEED PULLEY.

By H. C. SPAULDING. *Transactions of the American Institute of Mining Engineers, 1893, vol. xxi., pages 907-911.*

The device consists of a narrow pulley with a deep V-groove, split by a plane passing through the bottom of the groove, one of the two symmetrical parts so formed being fixed to the shaft, and the second being capable of adjustment by means of screws and handwheels, so as to bring the faces of the groove nearer or farther apart. By the use of a narrow or round belt, and with faces having the proper angle, the working radius may be varied within wide limits, while a loose sheave may be inserted between the two parts, thus combining the differential-speed principle with the equivalent of a fast-and-loose pulley. A. W. G.

PUMPING-PLANT AT BOCKWA COLLIERY.

Die neue unterirdische Wasserhaltungs-Anlage in Bockwa bei Zwickau. *By* C. SCHENCKE. *Jahrbuch für das Berg- und Hüttenwesen im Königreiche Sachsen, 1893, pages 1-40, with several drawings.*

This plant was designed to raise 3,300 gallons (15 cubic metres) of water per minute from a depth of 590 feet (180 metres).

The pumping-plant was designed in duplicate, each machine being capable of raising the average incoming water or 1,650 gallons (7·5 cubic metres) per minute.

The pumping-engines are placed in two chambers situated north and south of the shaft. The height of the engines is such that the water can accumulate for 6 to 8 hours in old levels before it rises to endanger the engines. There is also a connexion with a neighbouring mine below the level of the engine-rooms, and thus in case of emergency the water might be turned into the workings, so as to save the engines from being flooded.

The ventilation of the underground engine-rooms is secured by the end of the engine-room farthest from the shaft being connected with a special rise-staple and level leading into the upper part of the shaft. The air can consequently circulate, and an electrically-driven fan is placed in the shaft to facilitate the ventilation. The temperature in the height of summer has not exceeded 75 degs. Fahr., when the steam pressure was 90 lbs. per square inch.

The steam-pipes and boilers are designed to carry a pressure of 150 pounds per square inch (10 atmospheres). It was considered advisable to use high pressures, or at all events to be able to use high pressure in case of emergency. The steam-pipes, of which there are two, are made as small as possible, and the loss by condensation is found to be small. The pipes are of wrought-iron, with an inside diameter of 4 inches (100 millimetres). They are supported in the shaft at distances of 148 feet (45 metres), and at each support there is a copper expansion-pipe. This arrangement, for dealing with expansion, was not found to work satisfactorily, and it was found necessary to keep both steam-pipes filled with steam, in order to prevent leakages. The steam-pipes were tested by water to 300 lbs. per square inch, and the copper curved pipes to a steam-pressure of 120 lbs. The steam-pipes are guided in the shaft between two rollers. The pipes are jacketed with a preparation composed of cork, bound round with zinc wire, and

coated with gypsum, jute, and asphalte, and an outside sheathing of canvas, painted with Stockholm tar. The ends of the pipes are turned and bolted together with screw-bolts. The water-column consists of two cast-iron pipes 9¾ inches (250 millimetres) in diameter, and 29½ feet (9 metres) long. The pipes are supported in the shaft on iron girders and are guided between rollers. Expansion is provided for by means of bronze stuffing-boxes. The lower pipes were tested to 525 lbs. and the upper pipes to 300 lbs. per square inch. There are 4 Steinmüller boilers, each with 1,631 square feet (151·6 square metres) of heating surface. The boilers each contain 100 pipes, 3·7 inches in outside diameter and 16·5 feet in length, and one upper boiler 3·6 feet in diameter and 21·4 feet in length. The boilers are fed with a special fresh-water supply, the mine-water not being suitable. The engine-rooms, both at the surface and underground, are lighted by electric light.

Pumping Engines.—The engines were designed to pump the normal quantity of 660 gallons (3 cubic metres) per minute; and to be able in case of necessity to lift together 4,400 gallons (20 cubic metres) per minute. The engines are compound condensing: the high-pressure cylinder being 29·53 inches (750 millimetres), and the low-pressure cylinder, 43·31 inches (1,100 millimetres) in diameter, the length of stroke is 3·28 feet (1 metre); the average number of revolutions being 50, and the maximum 80 revolutions per minute.

The large plunger pump is 11·42 inches (290 millimetres), and the small plunger 7·87 inches (200 millimetres) in diameter, and the length of stroke, 3·28 feet (1 metre).

It was considered probable that in course of time the water would diminish, and that the pump would be required at the 1,287 feet level, consequently the parts of the engines were designed to resist the pressures at that depth, that is a pressure of 600 pounds per square inch (40 atmospheres).

The total cost of the installation amounted to £34,191 (683,832 marks), of which the two underground pumping engines cost £8,756.

The following results were obtained during the six months' run from January 1st to July 1st, 1893. Total water lifted, 146,660,800 gallons, or 562·5 gallons per minute, the costs for the same being as follows :—

	Expenditure. £ s. d.	Cost of lifting 1 cubic metre (220 gallons), 180 metres (590·4 feet). Pence.	Cost of lifting 1 cubic metre 1 metre high, metre-ton (7,216 foot-pounds). Pence.
1,350 tons coal, including cartage	472 10 0	0·1704	0·000960
Wages and superintendence	614 9 8	0·2208	0·001200
Lubrication	133 3 9	0·0480	0·000240
Packing	20 0 0	0·0072	0·000036
Repairs	6 0 7	0·0024	0·000012
Insurances, taxes, etc.	47 4 7	0·0168	0·000096
Timber, iron, and general stores	30 2 10	0·0108	0·000060
Totals	£1,323 11 5	0·4764	0·002604

W. F. W.

THE IMPACT PUMP.

A Paradox in Hydraulics—The Impact Pump. *By* GUS. C. HENNING. *American Machinist*, 1895, *vol. xviii., page* 5.

The impact pump consists of two parts: the piston, carrying an indiarubber valve, is driven by a connecting-rod through a stuffing-box in the usual way; the valve works in an enlargement of the suction-pipe, and the piston works in a cylindrical chamber below the enlargement. The rising-main is attached at one side of the valve-chamber.

The following table records experiments made upon an impact pump, whose piston was 3·156 inches in diameter, with stroke of 2·362 inches, and a displacement for one stroke of 18·46 cubic inches:—

Revolutions of Crank.	Work Transmitted. Q.	Orifice of Discharge above Water-level.	Actual Delivery.	Actual Work Done. Q^1.	Efficiency. $\frac{Q^1}{Q}$.	Theoretical Piston Displacement.
	Foot-pounds.	Feet.	Gallons.	Foot-pounds.	Per Cent.	Gallons.
295	27,510	49·85	46·70	20,950	76·1	23·5
302	29,047	49·85	54·78	22,820	78·5	24·1
349	32,547	49·85	55·25	25,324	77·8	27·9
352	30,810	49·85	62·38	25,985	84·3	28·1
360	31,590	49·85	63·32	26,380	83·5	28·8
417	37,536	49·85	75·19	31,325	83·5	33·3
278	41,442	101·7	46·07	39,146	94·5	22·2
289	44,828	101·7	47·34	40,225	89·7	23·1
299	46,434	101·7	51·45	43,721	94·1	23·9
308	48,170	101·7	53·02	45,068	93·5	24·6
319	47,953	101·7	51·30	43,577	90·9	25·5
321	48,388	101·7	53·02	45,068	93·1	25·6

The table shows that the capacity of the impact pump is more than double that due to the piston-displacement, whilst the efficiency rises from 76 per cent. with the low lift to 94 per cent. under the conditions of the high lift.

M. W. B.

PULSOMETER PUMPS.

Neuerungen an Pulsometern. ANON. *Dingler's Polytechnisches Journal*, 1893, *vol.* 290, *page* 1.

Amongst the advantages of the pulsometer pump may be counted its simplicity both of construction and working, the small amount of repair required, and the ease with which it may be fixed in inconvenient positions. The following are the results given of an experiment with a Körting No. 8 pulsometer pump on the Royal Hanoverian railway. The pulsometer was erected in the stream 65·6 feet (20 metres) from the boiler, and 22 feet (6·75 metres) above the level of the water; the water was raised to a level of 80½ feet (24·5 metres) above the pulsometer pump, so that the total height was 102·5 feet (31·25 metres):—

Duration of experiment ...	79 minutes.	
Quantity of water raised ...	144·46 cubic metres ...	31794·2 gallons
Water used in boiler	900 litres ...	198·0 ,,
Coal used	109 kilogrammes ...	240·3 lbs.
Temperature of stream ...	9·8 degs. Cent. ...	49·64 degs. Fahr.
Temperature of water in delivery-pipe	13·4 ,, ...	56·12 ,,
Number of pulsations per minute	57.	
Pressure of steam in boiler ...	7 atmospheres ...	103 lbs. per square inch.
,, ,, pulsometer	6·5 ,, ...	95·5 ,, ,,

Thus 402·16 gallons (1,828 litres) were lifted 102·5 feet (31·25 metres) per minute.

W. M.

THE CARE OF WIRE ROPES.

By J. H. JANEWAY, Jun. *The Colliery Engineer (Pennsylvania)*, 1895, *vol. xv., page* 148.

The author points out the economy of effective lubrication and recommends a first-class lubricant, which, he says, anyone can make, of tar, summer oil and mica axle-grease, in varying proportions to suit the conditions. The tar and oil must be free from acid. The combination is said to thoroughly penetrate between the wires to prevent rust, and fills the cable so as to give it the appearence of a bar of iron. It resists water successfully and does not strip off. Old wire-ropes which have been treated with this material, when taken apart, show every wire bright and clean, the heart or hemp core being in prime condition. As soon as a wire rope has been well filled with this compound, the addition of a very little, from time to time, is said to keep the rope in good condition, so that the lubricant is economical. It is stated that it costs in some cases only one-eighth as much per year as the old materials, with far better results; and, further, a well lubricated rope lasts from 25 to 50 per cent. longer time. J. W.

IMPROVEMENTS IN QUARRYING MACHINERY.

ANON. *The Engineering and Mining Journal (New York)*, 1895, *vol. lix., pages* 416-417, *and* 2 *illustrations.*

Two forms of channelling-machine, manufactured by the Ingersoll-Sergeant Drill Company, operated by compressed air or steam, are described and illustrated. The first drawing shows the improved stone channelling-machine doing corner and angle work in a quarry; the second shows the undercutting track-channeller for horizontal channelling.

The following figures, given by Mr. F. C. Neeb, a quarry operator in Ohio, show comparative results between hand, steam, and compressed-air drilling and reaming:—

By Hand.—Cost of drilling and reaming, September, 1891, 4,447 feet of top hole and 1,791 feet of side hole; total number of feet drilled, 6,238; cost 535·16 dollars; cost per foot 4·3d. (8·6 cents).

With Steam.—Cost of drilling and reaming, September, 1892, 5,629 feet of top hole and 1,874 feet of side hole; total number of feet drilled, 7,503; cost 438 dollars; cost per foot 3·25d. (6·5 cents).

With Compressed Air.—Cost of drilling and reaming, September, 1893, 8,007 feet of top hole and 2,394 feet of side hole; total number of feet drilled, 10,401; cost 449·50 dollars; cost per foot 2·15d. (4·3 cents).

These results include the cost of all blacksmith work, fuel, oil, etc. J. W.

A MINE RESPIRATOR.

Schutzvorrichtung gegen das Einatmen schädlicher Gase, insbesondere zum Schutz gegen Nachschwaden. ANON. *Glückauf*, 1894, *pages* 577-78.

This apparatus is intended to preserve miners from the effects of after-damp. It can only be used at mines in which compressed-air machinery is used, and consists of a long canvas sack, open at one end and closed at the other, which is connected by pipes with a compressed-air pipe. This sack is provided with a number of openings, fastened by strings. In case of need, the workman turns the tap which connects the sack with the compressed-air pipe, puts his head through one of the openings into the sack, tying it close round his neck with the strings, so that he can breathe pure air while the rest of his body is in the vitiated atmosphere of the mine. J. H. M.

WORKING OF SALT-MINES IN AUSTRIA.

Abschluss des Schachtsenkbetriebes am Ausseer Salzberg. By A. SCHERNTHANNER. *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1893. *vol. xli., page* 543

The author describes the results obtained by working the salt mountains in the Aussee district in Upper Austria according to his system. These mountains contain large salt-deposits, but the Salzberg is the richest of all. Owing, however, to previous defects in the working methods, the production had greatly diminished. Formerly the salt in this mountain was obtained by solution in horizontal layers by the action of water. An artificial lake was thus formed, flooded with water, and after due time allowed for impregnation, the brine was drawn off. The disadvantage of this system was that the water had an unavoidable tendency to spread: the workings thus became enlarged horizontally, and not only was the mountain undermined, but the brine continually diminished in strength. The different beds were broken through, and began to run one into the other, and the working results were negative. An attempt was made to remedy the evil by working two seams simultaneously one above the other, but to support them it was necessary to leave so great a thickness of rock that the loss was about 32 per cent. Owing to these reasons it was seen that the system of working the salt-mines was radically defective, and must be wholly remodelled, unless these rich salt-works were to be abandoned.

The faults of the method were, in the author's opinion, twofold. Firstly, the salt was worked from a horizontal area, whereas, if it were possible to work it vertically, the water would have the sides and roof of the excavation to play upon, as well as the bottom. Another disadvantage of this arrangement was that the residuum was gradually deposited on the bottom of the bed and prevented the free action of the water on the rock-salt. Each time the bed was flooded the water could not penetrate the *débris* and reach the rock-salt below. The net result was that the strength of the brine brought to the surface only corresponded to a proportion of 4·8 per cent. of salt. The second objection to the old system was that the water-flow could not be regulated. The water was first admitted at the bottom of the seam, and it was assumed that it would rise, gradually dissolving the salt above it, till the working attained a considerable height. Instead of this the water spread laterally. In one working, for instance, it was found that after the water had risen to a height of 13½ feet no more brine could be obtained.

The new method adopted was similar to that in use at the salt-works at Middlesborough. After sinking a small vertical shaft, an iron pipe was passed down it, and the flow commenced from the top, instead of the bottom of a salt-bed, the water being injected against the rock sides of the shaft. At its lower end the pipe terminated in a small jet. The water was first allowed to play upon the rock round the bottom of the shaft, and could be directed against it either horizontally or at an angle. As the space became filled with saturated brine, the action of the jet diminished, until a complete seam was excavated, which was then filled with water, and after a time the brine drawn off. The quantity of salt thus obtained was measured, and from it the size of the excavation calculated. As soon as, by drawing off the brine, the space had been enlarged to the dimensions previously determined, the injection of water was stopped. A calculated vertical thickness of rock above the seam was left as a support, and above it the jet was again introduced, and the process repeated. Thus a succession of spaces were formed one above the other, the dimensions of which were determined with mathematical precision from the volume of brine brought up. The lowest space was preserved intact by leaving it full of strong brine, and care was taken never to allow any of the seams to extend beyond the

bed of the original lowest working; their dimensions were easily regulated by the new method. To show to what extent the introduction of the water could be controlled, the excavations were sometimes carried right up to the top of the shaft.

One of the main advantages of the new system was that it permitted disused beds, which had been abandoned as unsafe, to be again worked. In many of them it had been found necessary to support the roof, but by the simple process of sinking a shaft over the roof, the props were rendered superfluous. By injecting water and drawing off the brine, the pressure of the roof was gradually removed, all danger of falling in was avoided, and the part of the mine above the old seam was made productive. The results of the new system were as satisfactory as the author had anticipated.

He divides the subject under three heads, and considers (1) theoretical results; (2) practical results under the old system; and (3) practical results under the new. Theoretically, with two beds worked after the old style, one above the other, to allow a depth of excavation of 42½ feet for the upper bed, it was necessary to leave a thickness of 16½ feet below and 19½ feet above, to support it. Practically, taking the beds yielding the most brine, when excavated to a depth of 121½ feet, the water only saturated the brine to a depth of 75½ feet, and the remaining 46 feet were lost. With the new system, saturation and excavation proceeded together. During a period of nearly 6½ years, the height of the column of dissolving liquid was 98 feet, and the quantity of brine produced 466,340 cubic feet per 24 hours. Comparing the two, about 86 cubic feet of brine were raised per hour with the old, and 188 cubic feet per hour with the new method, or more than double. The author attributes this striking result to the fact that the brine being drawn off from the bottom of the bed, while saturation takes place from the top, it is always carried off at a high pressure; and from the same cause the last quantity of brine is richest, and contains the largest proportion of salt. With a roof area of 32,280 square feet, about 10,590,000 cubic feet of brine are obtained. As this area may be doubled without risk of the roof falling in, the production of brine can be correspondingly increased.

The usual form given to the spaces is reversed in the new system. Instead of making the bottom broad and the roof contracted, to diminish the weight of the vault upon the sides, the roof is wide and vaulted. The walls are concave, narrowing to the bottom, which is often comparatively very small, but large enough to afford space for the accumulation of the *débris* and waste. B. D.

ALPINE SALT-WORKS, AUSTRIA.

Ueber die Ursachen der Todtsprechung Alpiner Laugwerke. *By* C. SCHRAML *and* AUGUST AIGNER. *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1893, *vol. xli.*, *pages* 555-559, 569-573, and 625-627.

It is a fact well known to salt-mining engineers that many salt-works, and those which appear most promising, fail suddenly, and often give way just when the heavy outlay on the plant, etc., is beginning to bear fruit. The attempt has often been made to examine more closely the causes of this phenomenon, and to explain them on theoretical grounds. The result showed that the existing methods of working in almost all Alpine salt-mines were very defective, chiefly because the saturation of the brine affected the stability of the surrounding rock. To obtain a clear view of the circumstances which caused the abandonment of many workings in the salt-mine at Hallstadt, Mr. Schraml has classified them in a tabular form, and this he was enabled to do with the greatest ease, because

for more than a hundred years the history of this mine has been carefully kept. It contains in all 124 workings, from the Catherina Theresia gallery downwards, which have at various times been abandoned, either because the brine was exhausted, or for other reasons.

There are four galleries or levels in this mine. The highest, the Catherina Theresia level, and the next the Leopold level, have been almost wholly worked out. The largest quantity of brine is at present obtained from the Joseph level, while the deepest of all, the Kaiserin Christina level, has hitherto been little excavated. In the Catherina Theresia level 64 workings have been abandoned; in the Leopold level 42; in the Joseph level 14; and in the Christina level 4. The mine was opened in 1681, and the first workings date from that period. The time during which brine has been extracted from these levels varies from four years (1819-1823, when the outlet from this working became unserviceable) to 172 years (1717 to 1889); in the latter case the workings fell in entirely. A startling fact is that out of 124 abandoned workings only 32, or one quarter, were exhausted, while in the remaining three-fourths other causes were at work to bring operations to an untimely end.

The causes of the failure of the different workings have been carefully analysed, and for this purpose Mr. Schraml has divided them into ten separate groups.

The first comprises those workings which have been abandoned because the brine was exhausted; the remaining nine fall under two heads, viz., three in which, according to the author, the causes of failure were avoidable, and six where they were unavoidable.

The second group include nine workings which have fallen in because the roof-area was too large, and several workings ran one into the other. Of these only four had a roof-area of more than 60,000 square feet, which used to be considered the bearing strength of a roof. In more than half, this limit was not reached. On the other hand, there are levels still worked where the roof-area of 97,000 square feet and more has remained intact for years; it seems therefore impossible to determine the limit of safety. In the case of four of these workings, where the roof-area was very small, their fall might have been avoided had the limits of resistance been rightly understood.

The third group comprises five workings abandoned through errors of construction.

The fourth group comprises ten small workings from which only a small quantity of saturated brine had been withdrawn when mining operations were stopped, and the floor and the rock turned to better account.

Thus far, workings abandoned intentionally, or from natural or avoidable causes, have been considered. The fifth group comprises those which have broken into the workings below, owing to fissures or porosity in the rock. One-fourth of the whole number belong to this class, and hence it is the most important of all. Is it, or is it not, possible to prevent workings in salt-mines from breaking down in this way? It must be remembered that brine is never so thoroughly saturated that an increase in its specific weight is impossible. Fissures in the rocks have been discovered containing brine which yielded 3·7 lbs. of salt per gallon. That brine can be so strong is owing to its capacity of taking up extraneous salts. In this way one may account for the otherwise inexplicable phenomenon that concentrated brine in the heart of the mountain eats away hollow vertical tubular spaces, often many feet in length. Thus one need not attribute the fall of the workings to actual fissures in the rock, which might seriously affect the present system of mining. There are numerous courses of anhydrite and other minerals impregnated with salts, which drain off the brine in one direction, and make it impossible to

raise it to the surface in another. Recently seven workings were rapidly abandoned one after the other, because the rock was porous and allowed the brine to filter away. Each of these had been formed on the Schwind system, one above the other, strong intermediate rock, 65 feet thick, divided the bottom of the level from the discharge-outlet, and some areas had already been worked successfully for years before the side walls reached the strata of porous rock.

In the sixth group the workings were abandoned because of the fall of the rock-roof. It is impossible always to fix the limits to which the salt extends. Often workings are opened containing abundance of salt, close to the periphery of the mine, and in the process of saturation and consequent extension of the space, the confines of the outer rock are reached. According to the character and stratification of the latter, it is either possible to continue working, the walls at the limit of the salt remaining vertical and intact, or if the roof of the spaces overhangs, the walls give way, the rock falls in, and the working must be abandoned. From the latter cause, no fewer than 16 workings have been closed. If to avoid this mischief, a broad zone were left untouched round the salt-deposit, the extent of which can often be only surmised, the number of workings in a mine would be greatly diminished, and the salt could no longer be produced at a profit.

The seventh group comprises workings abandoned through danger of cutting into other strata, levels, or shafts, and especially those in which the old method of solution had unduly widened the walls and rendered the dams or barriers useless. The new working-system of large iron pipes to admit the water, rapidly saturating the brine, and supplying water by a shaft or from above, will prevent this mischief in the future, especially if the dividing-barriers or dams be of sufficient strength. As these workings were abandoned because of erroneous methods of irrigation, their failure may be classed among avoidable errors.

The eighth and ninth groups consist of small workings abandoned either because during saturation beds of unproductive rock were encountered, or because the mineral became much poorer in salt.

In the tenth group are classed those workings where the rock above the salt strata fell in. Although sunk vertically one below the other, in accordance with mining rules, these workings had to be prematurely closed, probably because the roof was no longer able to bear the weight of the rock above it. In one of them, the supporting rock was 26 feet, in the others 36 feet thick or more, and thus according to received rules there should have been no danger of a fall. But new workings (also since abandoned) had been opened, the floor of the older levels watered, and thus their fall as it were actually prepared. The unequal supporting power of the roof was here most strikingly shown. While in many of these later underpinned workings the brine could be saturated without difficulty almost up to the working next above it, in others the intermediate rock fell in when brine to a depth of only 19 feet had been extracted, although the level above it was often not more than 32 feet in diameter. The heterogeneous structure of the rocks in this part of the mountain made it absolutely impossible to determine the bearing strength of the roof.

The following is a classification of the different groups of abandoned workings, according to their cause of failure:—32 workings, or 28 per cent., were worked out and exhausted naturally; 19 workings, or 17 per cent., were abandoned prematurely from avoidable, and 62, or 55 per cent., from unavoidable, causes; 11 workings fall under none of these divisions. In the workings prematurely abandoned, 77 per cent. were owing to the quality of the rock, and 23 per cent. to defective working methods. These figures may help to re-establish the reputation of the Alpine salt-mines, and explain their comparatively small yield of brine.

Hitherto, this has been attributed entirely to want of skill of the mining engineer, and the varying geological and physical conformation of the rock has not been taken into account. It is impossible to avoid the use of water to dissolve the salt, except by some dry process of working the mines, which would be far more costly. In spite of the better methods now employed to saturate the brine and support the galleries, Mr. Schraml fears that no considerable diminution in the number of abandoned workings is to be looked for.

From the same statistical information, Mr. Schraml also gives the number of years during which the workings in the different galleries are utilized, and the quantity of brine saturated. Of the four galleries in the Hallstadt salt-mine, the first two, containing respectively 61 and 45 workings, are almost exhausted. The average length of time for each working is 56 years in one, 82 years in the other, and the average depth of saturated brine in both galleries is 44 feet per working. The other two galleries are still being developed; there are 14 workings in the one, and 4 in the other, which have been utilized 47 and 50 years respectively. The depth of saturated brine is 23 feet. The average time during which a working is productive appears to be in proportion to the height of the roof, but the latter is independent of the quantity of brine obtained. The mean area of the roof in the 32 workings which were exhausted naturally was about 32,000 square feet.

The view taken by Mr. Schraml that the abandonment of the Hallstadt salt-workings is due, not to defects in the method of working, but chiefly to unavoidable causes, is contested by Mr. August Aigner. As early as 1849, Mr. Schwind laid down the important principle that between two workings in a mine there must always be a carrying wall, and it is owing to this condition that the Ischl salt-mines have been so successfully worked. Sixty years ago it was the general opinion that a yield of only 3 per cent. could be obtained from salt-mines. Mr. Schwind, who may be considered as the founder of modern scientific salt-mine engineering, set himself steadfastly against this view. To refute it, nothing, he declared, was necessary, but to adopt his own principles of working mines, which are now universally accepted.

Briefly stated, these were that the workings should be set one above the other, in tiers, like the stories of a house; and in spite of the expense involved, he considered it necessary to have a network of supporting walls from top to bottom of the vertical galleries. Had this condition been observed in salt-mining 50 years ago, the falling in of numberless workings would have been avoided. To prove, however, that even now it is not strictly adhered to, it is only necessary to examine the list given by Mr. Schraml of workings prematurely abandoned. Mr. Aigner has drawn up a chart representing a few of these in section, which shows at once that the excavations did not rest upon solid rock. The loss in working may be estimated by the fact that out of 124 workings, with a total height of 4,900 feet, to which the brine might have been saturated, it was dissolved only to a height of 1,920 feet, or only 40 per cent. A glance at the plan of the workings, with its irregular masses of intermediate rock, and the projections in every direction of the salt-bearing strata, always produced during the slow process of saturation, justifies the opinion that very little salt has been extracted, in comparison with what remains.

Mr. Aigner then proceeds to apply Mr. Schwind's principle to the Hallstadt mine. Here the galleries, being ranged one above the other, like the stories of a house, it would be vain to attempt to introduce a middle wall into this labyrinth of spaces. The whole of the mining region excavated is pierced with shafts for conducting water, which for hundreds of years has been at the same pressure in the shafts and galleries. The weight of all the workings has been estimated at

238,500,000 tons, and over them is a roof continually in motion, sometimes shaken by earthquakes, and resting upon no props, such as would be required in any building of the same extent. No wonder that the constant shifting and pressure of the loose mass of saliferous clay gives rise to all kinds of fissures, and that workings with a roof-area of only 5,400 square feet have to be abandoned, while others in a better situation can be worked in safety with an area of nearly 100,000 square feet.

It is true that we do not know all the influences which cause workings to give way, but by studying a few of those described by Mr. Schraml we may arrive at some understanding of the subject. Two of these fell in because another working was run below and between them, two others because they had been allowed to extend too far, and a fifth had to be abandoned because the neighbouring workings were unsafe. Another working gave way because it was situated above two others, one of which fell into a third, and the condition of the second was precarious. It is impossible to trace the causes of all the accidents in the 10 groups into which Mr. Schraml has divided the working of the Hallstadt salt-mine, but the foregoing are sufficient to show the futility of basing modern mining operations upon antiquated principles.

Mr. Schraml says, of 124 workings only 32, or one-fourth, have been naturally exhausted, and Mr. Aigner observes that this small useful percentage was obtained when, according to received opinion, the height of the galleries was only 92 feet, whereas on the modern shaft system they can be made 124 feet high. In the second group, 9 workings fell through owing to too great extension of the roof-area. One of these had an area of nearly 97,000 square feet; in the Aussee salt-mine there are workings which did not give way till an area had been attained of 146,000 square feet. The carrying power of the rock in a salt-mine is therefore enormous, and if these small workings had been surrounded with strong and indissoluble walls of solid rock, much better results might have been obtained. For every salt deposit there is a maximum roof-area that it can safely support, and the limits of which we can determine. The failure of this group must therefore be ascribed to the ignorance of the laws of stability prevailing in the last century.

Faults of construction and saturation of the floor are responsible for loss in the next 16 workings. Several of these would not have fallen in had they stood isolated. Thirty-one workings gave way owing to fissured or porous rock in the workings below them. This is attributed by Mr. Schraml to the fact that the brine, already saturated to the extent of 3·7 lbs. per gallon, became still heavier and ate its way down, by absorbing other salts; but few mining engineers will admit that this can happen. It would then be impossible to work salt-mines at all. In the Ischl and Aussee mines, which are as rich as those at Hallstadt in sulphates of soda, potassium, and magnesia, no case is known of one working running into another in this way. These solutions are more rapidly formed in fissures produced by general disturbance of the deposit, and hence by dividing the workings more regularly they may be avoided. The same may be said of cases where the workings were abandoned through the fall of the rock roof. In the Ischl mine, nearly all the workings towards the south have a sloping roof, and in only one has it given way.

In short, of the workings here examined all, except those which were naturally exhausted, failed because they were not properly supported. The method of saturating and drawing-off of the brine has already been brought to great perfection; but this will be of no practical use unless the working itself which yields the brine is maintained intact by a barrier of solid rock around it. B. D.

THE BRIXLEGG MINES, TYROL, AUSTRIA.

Der Bergbau Brixlegg im Tyrol. ANON. *Glückauf*, 1893, *page* 1437.

The Brixlegg mines belonged in the Middle Ages to the most celebrated mines in Europe, but now the industry is of only small importance. The lodes average 3 feet in thickness and contain principally silver-ore. The old workings were drained by the Bertha adit, but since 1881 a new adit some 200 feet deeper has been driven, and promises to be of great importance to the district. It has already attained a length of 3,000 feet.

The mineral output of the district will average annually :—

	Tons.
Ore for crushing, washing, and stamping	5,560
Copper and silver-ores saleable to smelting works	999
Iron-ores	1,006
Barytes	538

The barytes is ground fine : the annual sale of ground material being about 80 tons. The ore-concentration works are only worked in summer. W. F. W.

THE HIENDELAENCINA SILVER DISTRICT, SPAIN.

Hiendelaencina. ANON. *Revista Minera*, 1894, *vol. xlv., pages* 34-36.

The famous Hiendelaencina silver-mining district, which dates from the year 1844, is situated 65 miles from Madrid, 13½ miles from the station of Jadraque, on the Zaragoza railway, and at an elevation of 3,540 feet above sea-level. The deposits are concretionary lodes in gneiss, with a gangue of barytes, fluor-spar, and quartz, and bearing spathic iron, pyrites, and chalcopyrite. They continue perfectly characterized to the lowest depth yet reached (1,805 feet).

They are classed into three systems, one running east and west, with a northerly dip ; one north and south, with an easterly and westerly dip ; and a third with intermediate strike. Of these the first appears to be the principal ; it includes 3 parallel lodes, of which that known as the *filon rico* was the most productive in the early history of the district. Particulars are given of the mines now working on this and other lodes.

All the ore at present raised is treated by amalgamation at the Constante works, nearly 4 miles north of Hiendelaencina.

According to a paper written in 1875 by Mr. D. Miguel Bautista, the district produced up to 1870 a weight of 10,467,631 ounces of silver. G. E. C.

THE SILVER ORES AND OTHER MINERALS OF MILOS.

Les Minerais d'Argent de Milo. *By* L. DE LAUNAY. *Annales des Mines*, 1894. *series* 9, *vol. vi., pages* 345-353, *with map and section in the text.*

The general geological features of the island are as follows :—On the eastern coast are crystalline schists, overlain by trachytes, which are worked at Revma for grindstones and at Firlingos for sulphur. These in their turn are capped by tuffs, conglomerates, and highly fossiliferous Pliocene limestones. Along the southern coast range Jurassic and Neocomian limestones, dipping north, and rising in places to 1,600 feet above sea-level. Tertiary eruptive rocks, pyroxene sanidine-trachytes, form the chief mountain-summits of the island, and these are unconformably overlain by the liparite or quartziferous oligoclase-trachyte, which is essentially the metalliferous rock of the island.

The silver ores occur mainly as sulphides in association with barium sulphate in clay-pockets in the liparite; this clay appears to be a product of the surface-decomposition of the rock. Traces of silver chloride are present. Some of the masses of grey or black argentiferous clay are as much as 38 feet across. Generally speaking, the ores are poor, and where rich the percentage of silver is extremely variable. At Triades, the silver sulphide is disseminated in the clay in so fine a state of division that it cannot be differentiated by the naked eye. The workings have not been pushed very far; they consist of five small galleries and five shafts (the deepest of the latter being 72 feet). Twenty thousand tons of mineral were got out here between 1884 and 1890. At Cape Vani, the argentiferous deposits are closely associated with manganese ores, reminding the observer of a similar association in Montana and Colorada. Other silver-ore deposits have been discovered at Mount Mirovilia and at Pilonisi. The total mass of argentiferous rock in Milos has been estimated at 10,000,000 tons, with a mean content of 17½ ounces of silver per ton. The author quotes analyses by Mr. Cordella, and from the calculations made it would appear that ores of a grade lower than 21 ounces per ton cannot be exported with any advantage to foreign smelting-works. They must be treated on the spot in the wet way (chloridizing-roasting, lixiviation, and precipitation by copper) or at Laurium. The Hellenic Government have now taken in hand the working of the ores as a State enterprise.

Turning to the other minerals of economic interest, manganese occurs at several localities in the island, but the only important workings are those of Chalaka, Cape Vani (Miocene age), belonging to a French company. The number of workpeople is from 200 to 300, the wages being for miners 2s. to 2s. 9d. per day, for others 9½d. to 1s. 5d. per day. The annual production averages 18,000 tons, most of which is exported to Great Britain, France, and America.

Abundant sulphur-deposits, arising from solfatares, occur in pockets in the trachytes; the 2,000 tons yearly extracted are used chiefly in Greece itself for viticultural purposes. The State works a gypsum-deposit at Mount Damianos. Alum occurs abundantly, but so far has not been worked. A few brine-springs are known, and the salt from them is evaporated (260 tons per annum).

The grindstones of Milos have been in repute in the Levant for many centuries, and the quarries now worked near Revma belong to the State; the grindstone is a very compact yet porous silicified trachyte. O. S. E.

THE SILVER-LEAD ORES OF KARATSHAI, CAUCASUS.

Das Erzgebiet von Karatschai im nördlichen Kaukasus. By A. D. KONDRATIEFF. *Zeitschrift für praktische Geologie*, 1894, *pages* 369-380, *and plate VI., with figures in the text.*

The district of Karatshai covers an area of about 1,225 square miles in the northern Caucasus; it has only been opened up in recent years by the construction of a high road, and the mining rights were leased in April, 1894, for 90 years to the Elborus Company of St. Petersburg. Going southward up the Kuban valley towards the mountains, one passes successively over Tertiary marls and soft sandstone, Cretaceous, and Jurassic rocks, all dipping gently north-eastward and striking generally parallel with the Caucasus range. But the dip is seen to become much steeper as one approaches the mountains, and, near the old fortress of Chumara, greenstones (doubtless porphyrites) are met with, which account for the great disturbance of the stratified rocks: they are in part contemporaneous with the Jurassic. From this point onwards crystalline rocks predominate,

bearing only here and there patches of sedimentaries, and even these disappear altogether some 20 miles before one ascends to the peaks of Elburz. The core of the range is made up of several varieties of granitic rocks.

The Cretaceous strata offer little of interest to the miner, excepting the possible occurrence of rock-salt in the Neocomian. In the Jurassic are extensive lignite-deposits, mostly interbedded with sandstones; the thickness of the seams ranges from 1·8 to 6·5 feet, and at some localities the coal yields a very fair coke. Good fireclay occurs some 6 miles away from the silver-lead mines, and has been advantageously used for muffles and bricks.

The country-rock of the ore deposits is chiefly felsitic porphyry, curiously associated with talc-, mica-, and chlorite-schists of pre-Jurassic age. In this area of crystalline rocks also occur graphite-slates and iron-mica-schists with manganese ores.

Post-Jurassic movements of elevation of the earth's crust in Karatshai doubtless account for the system of fissures which now form the metalliferous lodes of that mining field. The author divides these into five groups, which are not only geographically distinct, but correspond with a particular facies, texture, and ore-combination of the lodes. These groups are :—

1. The *Indish Group*, the northernmost of any. Here three deposits are known, one of which (lower vein) is a combination of copper and lead ores, with 1·53 per cent. of metallic silver; the second (upper vein) is a combination of galena and iron pyrites, and the third consists exclusively of arsenical pyrites, an ore unknown elsewhere in the Karatshai district. A trial working showed that the gangue of the upper vein consists of ochre, calcspar, and fragments of the quartzitic country-rock.

2. The *Djalankol Group*, 3¾ miles south-east of the preceding one, comprises, so far as known, four lodes which are distinguished by their brecciated character; by the coarsely granular habit of the ore; and by the occurrence of heavy spar (here the chief gangue-material), from all other lodes in Karatshai. Three of the lodes attain a thickness of more than 6 feet.

3. The *Daut Group*, up the valley of that name, a tributary of the Kuban. Here seven lodes are known, one of which is notable for its large proportion of silver (0·257 for every 100 parts of lead). At the outcrop of these lodes weathering and atmospheric leaching have at many points destroyed all traces of ore, but on striking a few feet deeper down one generally meets with a thick metalliferous lode. The uppermost lode continues into a patch of Jurassic shales overlying the crystalline rocks, and thus gives a time-standard as to the epoch when the lodes of the district were formed.

4. The *Kuban-Chudess Group*. The map which accompanies the original paper shows that this is included in the triangular area defined by the course of the Kuban river and its affluent the Chudess. It is the best known and the most important of any group in Karatshai. The lodes are 3¼ to 5 feet thick, with smooth curved side-walls; the ores are galena and zinc-blende, with rarely a sprinkling of iron and copper pyrites, forming continuous seams in the lode. A disposition on the part of the galena to form a salband by accumulation at the sides is very noticeable. The gangue is made up of fragments of country-rock, and at some points the lode becomes a true breccia. More than forty separate ore-deposits are known in this group, and the author describes the most actively worked of these in detail. The saying of the German miners that a friable 'country' makes a poor lode is, on the whole, confirmed here. There are some exceptions, but the veins in the talc-schists are inferior in compactness and thickness to those

in the felsite-porphyry. The most important exception is the Chudess lode, which cuts obliquely through the eastern [? western] slope of that valley; it is about 9½ feet thick, and the country-rock is mica-schist. The absence of zinc-blende and other impurities heightens the value of this lode, as also the high silver content (0·17 in 100 parts of lead); in olden times it was worked by firing the rock.

5. The *Utsh-Kulan Group* comprises so far two lodes in granite-porphyry, one of which is highly argentiferous. This outcrop, high up on the hillside, allows of easy working by adits, and that is an observation which applies to lodes in the other groups.

The paper concludes with an elaborate table of all the known lodes in Karatshai, giving their exact localities, strike and dip, and mineralogical and chemical composition. O. S. E.

COLQUECHACA SILVER-MINES, BOLIVIA.

The Silver-Mines of Colquechaca, Bolivia. *By* ROBERT PEELE, Jun. *Engineering and Mining Journal (New York)*, 1894, *vol. lvii., pages* 78-79, 100-101.

Colquechaca, a mining town of some 7,000 population, is situate in Central Bolivia, in the eastern range of the Andes, at an elevation of 14,000 feet. It is about 90 miles by rough road, just constructed, from the narrow-gauge railroad which enters Bolivia from the Chilian port of Antofagasta.

Mining in the district dates back at least two centuries, but there are no records of production previous to 1865. Since then 21,000,000 dollars worth of silver has been extracted from the San Bartolomé tunnel alone, while 5,700,000 dollars has been raised since 1884 from the upper tunnel. From May 1st, 1892, to March 1st, 1893, the production was 809,000 ounces, which will be largely increased when the machinery now ordered is erected.

The deposits are fissure-veins in dacite and rhyolite, the most important being that known as the Embudo, which is traceable for over two miles. It dips into the mountain at from 75 degs. to nearly vertical. The ore-distribution is pockety and extremely irregular, but varies generally from 2 to 12 inches in thickness. The vein is clearly defined, with slickensides and clay gouge, and often contains cavities lined with crystals of pyrargyrite and quartz with wire silver. The rich ore consists mainly of ruby silver, native silver, and argentite, associated with sphalerite and a little galena; most of the value being in ruby silver. It runs from 500 to 5,000 ounces, second-class ore varying from 100 to 200 ounces per ton.

The other lodes of the group lie on one side of, and within 400 feet of, the Embudo. They are similar in character and hold good in depth, but their lateral extension is small.

The hoisting and pumping arrangements are bad and inefficient, at one mine only 20 per cent. of the underground hands being engaged on productive work; but this will be modified consequent on a consolidation in May, 1892, by which all the mines are placed under one management. The San Bartolomé tunnel has now been straightened throughout its length of 5,035 feet to permit of the use of locomotives.

The high-grade ore is picked, hand-jigged, and exported to Europe. The ores under 300 ounces per ton are treated at primitive amalgamation-works near Colquechaca, where the loss in tailings averages 30 per cent. of the assay value. The charge for treatment was formerly about 60 dollars per 2,000 lbs., the actual costs being somewhat as follows:—

	Dollars.
Crushing	2·25
Roasting	7·25
Barrel amalgamation	9·55
Retorting, superintendence, etc.	6·50
Total	25·55

Amalgamation in *fondos* (copper-pots, set on rude furnaces) cost about 10 dollars per ton more.

One of the principal difficulties has been the enormous amount of ore-stealing by the Indians, who are remarkably ingenious in this respect. It is hoped that this will be materially lessened, as men once found out will be unable to get employment elsewhere, as was formerly the case. G. E. C.

METALLURGY OF SILVER IN CHILE.

Metallurgische Beiträge aus Chile. Das Auftreten der Silbererze zu Challacollo und deren Laugung zu Cerro gordo. By ANDREAS GMEHLING. *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1893, *vol. xli., pages* 468-471 *and* 479-483.

The mineral district of Challacollo lies at the foot of the Cordilleras, about 200 miles east of Patillas. The isolated range of Challacollo, rising about 1,000 feet above the Pampas, consists principally of quartz-trachyte. The principal veins run from south to north, and are cut through almost at right angles by subsidiary veins; the ore is usually richer at the point of junction. It is found as dry silver ore, and horn or chloride of silver, with a small proportion of barytes and lime. The barytes occurs as heavy spar, the lime chiefly as calcspar and gypsum. Lead sulphide or galena is found in bunches, but the quartz ore also contains lead carbonate, chloride, etc., which are rich in silver. Zinc oxide, carbonate and silicate of zinc, iron oxide, alumina, manganese, etc., are also present, impregnated with 10 per cent. of copper, and containing much silver and sometimes gold. The presence of iodine and sodium chloride or common salt, saltpetre, sulphates of magnesia and of iron seems to prove that the silver in the veins has been separated chiefly by a solution of common salt.

The mines have been sunk to a depth of 426 feet, and so far have not shown any variations in the character of the ore. At present only the six mines belonging to Messrs. Sotomayor, Carrasco, & Co., are commercially valuable. They contain four principal veins, those which crop up at the surface being richest in silver. The veins run from south to north with a dip west of 25 to 27 degrees, and thin out towards the south, where the main shaft is sunk. Better ore is found further north, and they are here concentrated at the surface for about 328 feet; as the ore is richer where the veins run together this vein has till now proved the most productive of any. The lodes are reached by galleries and shafts. The vein of ore is from 13 to 23 feet wide, and contains from 8 to 18 per cent. of silver. It is richest on the eastern side, where it is intersected by two cross veins running east and west, and forming a nucleus containing much silver. Towards the end of this vein a second shaft has been sunk. Throughout the mine, the vein of ore contains most silver on the east side, and is usually but not always richer where traversed by cross lodes. It never disappears, and is generally of considerable thickness, but its proportion of silver varies greatly, according to the inclination of the strata, and the occurrence of cross veins. To a depth of 262 feet, the principal vein is

nearly uniform, but it is then cut through by a quartz-trachyte cross lode, beyond which it is looser and not so compact. Wherever the ore is found in this loose and broken condition, it is poor in silver, containing only from 2 to 3 per cent. Below this, four veins can be distinctly traced, about 25 feet wide, more solid in composition, and with from 4½ to 6 per cent. of silver.

Extraction of the Ore.—The dry silver ore has a mean content of 10 per cent. of silver, principally in the form of chloride and iodide, and 0·0001 per cent. of gold. The chief silver-bearing minerals in the vein are:—Lead carbonate and galena, lead chloride, and a mixture of these with lead sulphate. Although the greater part of the silver is present as chloride or horn-silver, the ore is too compact to be treated in a raw state. To break it up and loosen the silver compounds a small quantity of salt is added, and the ore is then roasted in a revolving cylindrical kiln.

The composition of the ore fluctuates; during 1892 it was as follows:—Silver 9 to 12 per cent.; gold 0·00010 to 0·00015 per cent.; silica 75 to 84 per cent.; other minerals, such as lead, copper, zinc, manganese, in quantities varying from 7 per cent. to small traces, with a very minute percentage of moisture. It also contains rock-salt, saltpetre, sulphate of aluminium, iron, lime, magnesia, etc. In one sample these amounted to 4·25 per cent. Only about 7 to 12 per cent. of the silver and lead chloride can be dissolved in hot water, but much more is obtained with a 10 per cent. solution of cyanide of potassium. Of the horn-silver, 8·26 per cent. is set free in hot water; about 9·87 per cent. in a strong hot solution of salt; from 12·23 to 55 per cent. in solutions of hyposulphite of soda; 43·82 per cent. in ammonia solution, and 77·53 per cent. in a 10 per cent. hot solution of cyanide of potassium. The quantities vary according to the time allowed for dissolving, the best result being obtained after saturating the ore for 24 hours, and the lowest after 72 hours. By the process of amalgamation, from 60 to 70 per cent. of the silver can be extracted.

The ore from the Buena Esperanza mine, about 30 miles from Cerrogordo, is delivered in pieces of three sizes, the largest containing about 1 per cent., and the smaller sizes 8 and 6 per cent. of silver. There is no water in the mine, nor does it require any masonry or woodwork, the rock being firm, and the few props left here and there are scarcely necessary for safety. The ore is brought to the surface by a horse-gin; the working, raising, and separating costs from 8 to 9 pesos per ton. (The Chilian peso is constantly changing in value. At the present time, 1893, it is worth about 1s. 4d.; its nominal value is more than 4s.) The ore is then conveyed to Cerrogordo on carts drawn by mules, and holding about 2 tons each; the cost of transport is about 9 to 10 pesos per ton.

The larger pieces are broken small by hand, and are then delivered with the smaller to a ball crushing-mill, three of which are used at the mine. Each holds about 13¾ cwts. of cast-iron balls, makes 22 to 24 revolutions per minute, and delivers per hour from 17 to 27 cwts. of ore crushed fine enough to pass through a sieve. To grind this quantity 9 horse-power are required. Salt is fed in with the ore in the proportion of 8 per cent. It is next conveyed to rotating cylindrical kilns (Howell system), raised by an elevator into the kiln-hopper, and shot out automatically at the further end. The inclined cylinder is 27 feet long; it revolves half to one revolution per minute, and the ore is roasted in about an hour. From 98 to 107 cwts. of wood is required for firing per 24 hours, or from 49 to 53 cwts. of Australian coal. The wood costs 0·6 peso per cwt., and the coal 1·30 pesos per cwt., but wood is preferred, because combustion is more regular, and the walls are not burnt. It is found about 12 miles from Cerrogordo, covered

with sand to a depth of 3 feet. 16 pesos are paid on the spot per waggon-load of 29 to 35 cwts. of wood, and from 30 to 40 men are employed in digging it out.

Each furnace roasts in 24 hours from 686 to 744 cwts. of ore, which then falls into a brick chamber below, where it stands for some hours and becomes thoroughly chloridized. During roasting the ore swells and diminishes in weight and in proportion of silver, the loss depending chiefly on the temperature of the furnace. At a low temperature, the average loss in weight is 5·80 per cent., in silver 0·30 per cent.; and at a clear red heat, 10·40 per cent. and 31·82 per cent. respectively. To add more than 8 per cent. of salt to the ore is of no advantage. The ash is from 2 to 3 per cent., and contains almost the same proportion of silver as the ore, to which it is gradually added. One fireman is required, 1 man to bring the wood, and 1 boy to work the elevator. After roasting, the ore contains from 8 to 12 per cent. of soluble salts, and is best treated with a solution of 1 per cent. of hyposulphite, which extracts not only the silver, but whatever gold is present.

The crushed and roasted ore is next dissolved in troughs, 12 feet broad, 14 feet long, and 2¾ feet high, each containing a wooden filter covered with sacking, and holding 7½ tons. The ore, hot from the furnace, is poured into the troughs, and covered with water, which begins to boil immediately, and takes up the greater part of the soluble salts, principally common salt, and lead and silver chlorides. The height of water above the ore is 4½ inches, and about 247 cubic feet are needed for washing the ore the first time. As soon as the liquid ceases to boil, a plug below the filter is removed, and 153 cubic feet of water are run off, 94 cubic feet being absorbed and evaporated. The water discharged is at a temperature of 172 degs. Fahr. From 1½ to 2 hours are required for filtration.

The quantity of soluble salts dissolved in 1 litre (61 cubic inches) of water varies considerably; in the trough mentioned above, it was 4·3 ounces, with 7 grains of silver, or 65 ounces silver in 153 cubic feet of water. If the hot liquid be allowed to cool, the lead and silver chlorides fall to the bottom, being deposited in the same way as on the floor of the trough. The ore is then soaked in 159 cubic feet of fresh water, which take up 9½ ounces of silver; two hours are needed for this second process. All the water is then run into another trough, and the silver precipitated with sulphide of sodium. A considerable economy could be effected if the water were allowed to cool, and the silver naturally deposited. But this would require two days, and the quantities dealt with are so large that the simpler plan of precipitating the metal with sulphide of sodium has been preferred. The operation takes 1½ hours, and the sulphide obtained contains from 6 to 10 per cent. of silver. If time permitted, most of the lead, silver, copper, and iron could be separated from the water; but from 10 to 15 per cent. of the silver is generally left. Sulphide of sodium has been found the best and most economical precipitant, because all other methods entail loss in the extraction of the silver.

The ore remaining, after the water has filtered through it, is then dissolved six times in a solution of ½ per cent. of hyposulphite of soda. The separation of the silver chloride takes about as long as filtering. The first two infusions yield 102 ounces of silver; the next two, 34·8 ounces; the last two, 10¼ ounces of silver. Dissolving the ore thus takes 15 to 18 hours, the whole process, therefore, from the time the hot ore is first placed in the troughs occupies from 22 to 24 hours. The quantity of silver obtained from one trough holding 150 cwts. is as follows:—In the water, 76 ounces, or 30·65 per cent.; dissolved from the ore, 146·6 ounces, or 59·22 per cent.; total, 222·6 ounces. The ore contains 247 ounces of silver, therefore 24·4 ounces are lost, or 10·13 per cent.

The hyposulphite solutions are precipitated with sulphide of sodium. The deposition of the sulphur or metallic sulphide, containing 25 to 30 per cent. of silver and a little gold, takes 1½ to 2 hours; it is then dried and exported to Europe. Silver is also obtained by burning the old filters and sacks, the ashes of which contain from 8 to 12 per cent. of silver and a little gold. Formerly the wood and sacks, although impregnated with silver, were used to fire the boilers; 875 pounds of refined silver were obtained from them in the last half of 1892.

The sulphide having been precipitated, the hyposulphite solution is drawn off and pumped back to the vats through a layer of small ore, and in 2 or 3 months this ore absorbs from 4 to 8 per cent. of silver; it is then exported to Europe. In whatever way the ore is treated there is always a small unavoidable loss of silver, which has lately been considerably reduced, owing to the careful sifting of the refuse. In the latter half of 1892 about 5,106 tons of roasted ore were treated, with 9¼ per cent. of silver. In round numbers, the loss during precipitation may be taken at 12 to 14 per cent., but this is, of course, less, and the output more valueable, in working richer ore with 15 to 20 per cent. of silver.

The hyposulphite solution loses its strength during the working process by the decomposing influence of the air, the partial decomposition of the liquid, its dilution with water and saturation with other salts. The strength of the solution is therefore tested with iodine every day, and never allowed to fall below 4 per cent. For every ton of ore, from 2½ to 2¾ pounds of hyposulphite are required. The troughs in which the silver is precipitated are of the same size as the filtering-troughs.

The sulphide left at the bottom of the troughs is washed, dried by the heat of the sun, and exported to Europe. The mean proportion of metals in this *sulfuros* or solid sulphide is:—Silver, 17 to 22 per cent.; gold, 0·010 to 0·018 per cent.; copper, 6 to 12 per cent.; lead, 10 to 20 per cent.; zinc, 5 to 12 per cent.; sulphur, 25 to 30 per cent.; and soluble salts, 13 to 16 per cent. Part of the sulphur can be separated by heating the metal with a solution of caustic soda, and the sulphide of sodium thus obtained may be again used to precipitate the silver.

The entire process of treating the ore after it has been brought to the surface costs from 18 to 22 pesos per ton. The price of silver has now (1893) fallen so low that it is just possible to make a small profit on ore containing 8½ per cent. of silver, if 8 troughsful are treated per day, or 833 to 930 tons per month. The cost of production depends also on the value of the Chilian peso; if the latter were suddenly to rise till it was worth about ¾ ounce of silver, all metallurgical operations would necessarily have to be suspended.

Workmen's wages are high in this district, provisions and other necessaries are also very dear and much more expensive than in Europe. All these articles are supplied by the company working the mine, who keep a store and realize an average profit of 100 per cent. An ordinary day-labourer earns 3 pesos per shift; superior workmen, 4 pesos; foremen, 5 to 7 pesos. Mechanics, smiths, and carpenters are paid from 150 to 200 pesos per month; masons, 6 pesos a day. The head mechanic receives 250 pesos per month and his lodging; his assistants, 4 to 5 pesos a day; and the firemen the same. Piece-work is usual throughout the mine, and a skilful miner earns 120 to 150 pesos per month.

A list of prices of food and clothing is given in the original paper.

B. D.

THE ORE-DEPOSITS OF CREEDE, COLORADO, U.S.A.

By THOS. R. MACMECHEN. *Engineering and Mining Journal (New York), 1892, vol. liii., pages 301-303, with three ink photos of the Jimtown district.*

The author treats the subject under the following headings:—Geological formation, nature of the ore-deposits, the Last Chance-Amethyst vein, the Holy Moses vein, and Mammoth Mountain mines.

Referring to the Last Chance-Amethyst vein, he says this feature of the camp claims precedence because of its value at so early a stage of development. Along the surface, for a distance of over 1,100 feet, it has been exposed to a depth of 150 feet by drifts on the vein connecting the entire body, and during the time of exploration not more than 4 tons of waste rock were extracted. This development showed the existence of a vein of an average width of 6 feet. The average grade of the ore-body is from about £16 to £32 per ton. Upon a cautious estimate, says the author, about £300,000 worth of ore is in sight above the 100 feet level, and the heaviest body in the Amethyst vein is displayed in the north drift. Below the 100 feet level and between the two shafts, located about 200 feet apart on the vein, from £160,000 to £200,000 worth of ore is expected to be mined. J. W.

THE ORE-DEPOSITS OF CREEDE, COLORADO, AND THEIR POSSIBILITIES.

By EDMUND B. KIRBY. *Engineering and Mining Journal (New York), 1892, vol. liii., pages 325-326.*

The author had examined the geology and ore-deposits at this silver-camp in the latter part of September, just before the heavy snowstorm stopped geological surface-surveying for the winter.

A description of the geological features is given, also of the veins. The quantity of silver in the large bodies (8 to 20 feet thick in the Holy Moses vein) is said to be 10 to 15 ounces per ton, but large streaks and patches through the mass are high grade, bearing 50 to 100 ounces per ton, and these constitute the shipments sent to market. This pay-ore is distinguished from the mass only by assay. It is nearly free from lead and gold. A quartz streak occurs, but is said to be low in silver, so the mining is mainly the development of the high-grade ore streaks in the soft hanging-wall.

The present rate of shipment from the district is 150 to 175 tons of ore daily; 100 tons of this being from the Last Chance, and the rest from the Amethyst, Holy Moses, and Ethel mines in the order named. This, says the author, is a remarkable record, and justifies large expectations for the future.

The short period of five months, in a winter with heavy snowfall, has seen the advent of the railroad, the shipment of some 8,300 tons of high-grade ore, and a rush to the district that has increased the population from 200 to 7,000 men.

J. W.

THE BROKEN HILL SILVER-MINES, NEW SOUTH WALES.

By T. A. RICKARD. *Engineering and Mining Journal (New York)*, 1891, *vol. lii., pages* 530-532.

Broken Hill, reached by 300 miles of railway from Adelaide (South Australia), but inside the New South Wales border, is in the Barrier Ranges, which were known as a rich but patchy mineral-field long before the discovery of the silver deposits which have lately made them famous. It was a cowboy or "boundary rider" who first discovered silver where he thought a tin-mine might possibly be commenced. This was in September, 1883, since when the said cowboy and his fellow-stockmen have become millionaires. The development of the mines has been carried on till an output of from 175,000 to 200,000 ounces of silver per week has been attained.

As seen from the mine, the surrounding country is of dreary flatness, broken only by occasional low hills. The outcrop of the lode forms a black ridge of manganic ironstone, whose irregular contour gives the name of Broken Hill. Close at hand extends the town of Broken Hill with its population of 30,000.

Connected with the big mine—the Broken Hill Proprietary—are 13 furnaces, the output of this plant being a ton of silver per day.

The following figures give the annual yield of the Broken Hill field up to the close of 1890:—Prior to December, 1887, 10,378,883 ounces; for the year 1888, 4,020,370 ounces; 1889, 6,358,463 ounces; 1890, 7,824,605 ounces.

The Broken Hill Proprietary mine is the largest and has the largest output. Up to the date of the last report, 654,873 tons of ore had been treated, producing 25,729,834 ounces of silver and 107,037 tons of lead, having a total value of £5,893,722.

Up to the close of last year the average value per ton of ore was £9 7s. 7d.; the average cost £4 7s. 9d., and the average profit £5 1s. 10d.

The mine is worked by means of eight principal shafts, the bulk of the stoping so far done being confined to that part of the mine which is above the second level.

The lode as seen at the No. 2 level, whose average distance from the surface is 215 feet, is of extraordinary size and value.

More so perhaps than any other modern silver mine, the Broken Hill Proprietary is enriched by a varied assortment of silver-bearing minerals scattered in splendid profusion through the ore. Block XI. is especially remarkable. There is there an ore-body 270 feet long, averaging 200 feet in width and carrying from 50 to 55 ounces of silver per ton, together with 30 per cent. lead. The candle-light shows large faces of manganic ironstone studded with the half-formed crystals of iodide and chloride of silver, the two being readily distinguishable by their colour. In the third level, the sulphide ores are of low tenor and a very refractory character, containing for the most part from 16 to 18 ounces of silver, and 25 to 30 per cent. of lead, mixed with a large percentage of blende and occasionally pyrites. The sulphides are loose and friable, the coarser crystalline galena is richer than the fine-grained, and carries as a maximum 60 ounces of silver.

The great question which arises sooner or later in so many silver-fields, that of the replacement of easily reducible oxidized ores by more or less refractory sulphides, has already arisen at Broken Hill. Galena, blende, and garnetiferous sandstone form ore of a discouragingly refractory character. When its silver contents are low the question of its economical treatment is one which will try the resources of the ablest metallurgist. At the present time strenuous attempts are, with the enterprise characteristic of the company, being made to solve the problem.

J. W.

THE EICHLER DOUBLE WATER-TIGHT PILING.

Neue hohle doppelwändige, glatte und wasserdichte Spundwand. ANON. *Glückauf*, 1895, *vol. xxxi.*, *page* 931, *and* 6 *figures in text.*

The Eichler piling system is suitable for lining circular or multi-angular shafts or galleries in water-bearing strata, and keeping out loose sand, for excavating for foundations or bridge-piles, especially where ramming or shocks must be avoided; in breakwaters or sea-walls, either as a protective work or as a wall filled with concrete; and if perforated and the internal space filled with gravel, makes an excellent filter-casing for wells.

The materials employed in its construction may be either wrought or cast-iron, or a combination of the former with wood. Each section or pile is formed of two similar pieces, about 12 feet long (either of iron plate fitted with an iron bracket, or of a single piece turned up to form an angle), placed side by side, and fastened together by stayed bolts. A completed section has flat surfaces on both inner and outer sides, and by means of a groove left open at both ends, it can be interlocked with the corresponding portions of the adjacent sections, the edges fitting into suitable recesses, and thus forms a strong joint, which external pressure only assists to preserve. The iron-plate employed varies from $\frac{1}{32}$ to $\frac{1}{2}$ inch in thickness, 12 feet in length, and about 10 inches wide, with suitable iron brackets placed at one side. Two sections, when placed together, may have an external thickness of 3 to 4 inches or more, according to requirements. Wooden piles are cheaper than iron, and are made of planks $1\frac{1}{4}$ to $1\frac{1}{2}$ inches thick, but the dimensions, both of these and of the iron piles, can be increased, if desired, by using thicker plates, or by bolting several lengths together. The lower end of the section-plates may be protected with iron-shoes if necessary, but usually filing down the plate to a feather-edge will suffice.

The fixing the piles in position may be effected by means of a ram, jack, or hydraulic press, according to the nature of the ground, the sections being placed in juxtaposition and driven in, although not at equal depths, until the work is completed. C. S.

METHOD OF PLUMBING SHAFTS.

By A. NEUSTAEDTER. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxi.*, *pages* 792-794.

The writer here describes a method of plumbing shafts which he has used with good results. It consists in obtaining a longer base by suspending the plumb-lines from the extreme corners, lengthwise of the oblong shaft, and reproducing the bearing by offset-points on sticks attached to plumb-lines and supported on floating corks. A. W. G.

METALLURGY OF IRON PYRITES.

Die Kupferdarstellung aus gerösteten Kiesen. By ROB. SCHELLE. *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1893, *vol. xli.*, *pages* 517-522 *and* 531-534, *and* 2 *plates.*

Sulphuric acid, which till within the last 50 years was extracted from sulphur only, is now chiefly obtained by roasting pyrites. Treated by the dry process, the pyrites was useless unless rich in different metals; now the raw material represents a valuable product, even if it contains nothing but iron. At first, copper pyrites was roasted to produce sulphuric acid, and the copper and other

metals were separated by smelting. As the price of sulphur rose, iron pyrites was used for its extraction, and a much larger residuum, containing iron, copper, and silver, remained after roasting to be dealt with. This could not be treated by the dry process of smelting, on account of the expense. The wet process was therefore adopted, namely, the separation of the copper, silver, and sulphur in the roasted pyrites by dissolving with water and salt, and the subsequent utilization of the iron.

The first experiments were made in England, and the method there introduced is now universally employed. There are various places in Germany and Austria where iron pyrites after roasting for sulphuric acid is treated in this way, sometimes native, sometimes Spanish pyrites being used. The composition of the pyrites varies greatly, especially in its proportion of copper. Spanish pyrites is richer than Hungarian, but never contains more than 3 to 4 per cent., ore with a higher percentage of copper not being exported. The poorer pyrites is sent over in shiploads to the sulphuric-acid works in Germany and England, and thence to the copper-refineries. Hungarian pyrites contains less copper and more silver, iron, and sulphur than the Spanish. The proportions are:—In the Hungarian, 40 per cent. of sulphur, 40 per cent. of iron, ½ to 2 per cent. of copper, and 0·0055 per cent. of silver. In the Spanish, 48 per cent. of sulphur, 40 per cent. of iron, 4 per cent. of copper, and 0·003 per cent. of silver. Two million tons of pyrites are brought annually from Spain and England, and 30,000 to 40,000 tons are obtained from a mine in Hungary.

In the roasted products, or pyrites from which the sulphur has been extracted, iron is present as oxide, copper as a metal, oxide, or sulphide, which cannot be dissolved in water only. To extract it the following processes, mostly mechanical, are employed:—Pulverizing, roasting, lixiviating, or impregnating and dissolving with salt, separating the silver and copper, and working up the precipitated copper.

As the roasted pyrites is brittle, it is easily pulverized, and must be crushed fine enough to pass through a sieve with holes ⅛ inch square. Salt is added, the quantity varying according to the percentage of copper and sulphur. A certain amount of sulphur is necessary, otherwise it is impossible to chloridize the copper; the proportion should be about 1½ times that of the copper. The proportion of salt is from 7 to 20 per cent.

The pyrites is then roasted, to render the copper soluble in water, or in diluted sulphuric or muriatic acid. Some pyrites, especially Portuguese, is so brittle, and the copper so equally distributed throughout, that no roasting is necessary. It is sufficient to soak it several times in the original solution, until the copper is dissolved. Usually, however, the pyrites is too solid to be treated thus, and must be roasted with salt. With ordinary roasting and a low furnace-temperature, more copper is dissolved, and the solution contains more iron; at a higher temperature, the copper is not completely dissolved, a large quantity of sulphur is left in the pyrites, and iron cannot be obtained from it. But if salt be added, and the pyrites instead of being oxidized is chloridized by roasting, a different and much more favourable result is produced. A high temperature is then of advantage, and the greater part of the copper may be extracted, with very little iron. Part, however, is transformed into insoluble oxide and chloride, and therefore the higher the temperature of the furnace during roasting, the larger the quantity of copper obtained.

At first ordinary furnaces were used, but were found unsuitable, because the temperature was not uniform, and the gases of combustion mingled with the

chlorine and hydrochloric acid vapours, and made them difficult to condense. In the two types used chiefly in Lancashire and Hungary, the gas flames are first introduced below the furnace charged with the pyrites, and then carried over it, the air necessary for combustion entering separately. Two and a half tons of pyrites can be treated in 24 hours in one of these furnaces. Their disadvantage is that the hydrochloric acid gas generated becomes mixed with the gases of combustion, and can with difficulty be condensed. Vaulted furnaces have therefore been adopted in the larger Austrian chemical works. In these the flames do not come into contact with the pyrites, but the latter are heated in a separate chamber.

A furnace of this kind has in the centre a vaulted space, the floor of which, charged with pyrites previously dried and heated, rests upon the double grate. Air is separately admitted. The flames are carried first above, then below the vaulted chamber, and the products of combustion led off into the chimney. The gases generated from the pyrites during roasting are conveyed to a condenser. These furnaces require more fuel than the others, but as the temperature is low, an even heat is maintained, upon which the extraction of the copper depends. Other furnaces, mostly used in England, combine the two systems. The flames do not come into direct contact with the pyrites, but pass over a wall built half way across the furnace, the roof of which slopes upward. The best type has a circular rotating iron floor, lined with firebrick, and is fixed in the bottom of an ordinary vaulted furnace. The combustible is fired in a separate grate at the side, but the flames pass on to the circular floor, on which the pyrites is placed, mingle with the gases generated, and are carried off to the condenser. The floor is fixed on a vertical axis, and is rotated by means of an endless chain. The pyrites is mixed, distributed, and withdrawn from the furnace by machinery. The air necessary for combustion is separately introduced. At the Bede metal works, 12 of these furnaces are in use, worked by a 12 horse-power steam engine, and all the parts except the mixer last a long time.

With an ordinary furnace the charge of pyrites is from 2 to 2¼ tons, and is spread over the floor to a depth of 4 to 5½ inches. The furnace is then heated to a dark red, the pyrites being kept continually mixed, the roasted mineral periodically withdrawn, and the heat maintained uniform. Blue flames appear, and after about 5 hours, the burning pyrites becomes of a greenish-grey colour, and the process of roasting is complete. The charge is then emptied into the troughs, and the copper contained in the pyrites dissolved in water; 75 per cent. can be thus obtained, 20 per cent. dissolved with muriatic acid, 5 per cent. with nitrate, and 5 per cent. is usually wasted. The time of roasting varies with the amount of sulphur and copper in the pyrites. To roast poor pyrites in a revolving furnace takes 6 hours; rich pyrites in ordinary furnaces, 10, 12, or 15 hours. Cheap coal is used for firing. In the Duisburg metal works there are 12 vaulted furnaces, in which Spanish pyrites are roasted 8 to 10 hours with 15 per cent. of salt; each charge weighs 2½ tons. At Hemixen in Belgium, 5 tons of Swedish pyrites are roasted at a time for 6 hours, with 20 per cent. of salt, and 170 lbs. of fuel are burnt per ton of pyrites. The temperature of the furnace varies from 752 to 932 degs. Fahr. Sometimes from 5 to 7 per cent. of sulphate of iron is added to the salt, and reduces the time of roasting by one-third. It is said that 96 per cent. of the copper can afterwards be dissolved out with water alone, and no hydrochloric acid is necessary. Of the copper, 3·25 per cent. is obtained if the pyrites be roasted in the ordinary furnace, 3·53 per cent. if roasted in a revolving furnace.

The gases are condensed in the same way as the gases from sulphur furnaces used in the manufacture of soda. They are led into isolated towers of brick,

bound with mortar made of tar and sand, and filled with coke or firebrick, over which water runs continually to absorb the hydrochloric acid gas. Smaller towers may be used with vaulted furnaces, and condensation is said to be more rapid if the walls are thin. Most of the acids thus obtained are required to dissolve the chloride of copper from the pyrites.

The roasted and chloridized pyrites is then heated in troughs and lixiviated, or dissolved out with salt and water. The troughs, 10 to 13 feet long and 5 to 6 feet deep, are of wood, lined with tar or asphalt, with filters at the bottom. The pyrites is first soaked in cold water or a weak copper solution, then twice dissolved in boiling water. The solution thus formed contains all the copper compounds soluble in water, and 95 per cent. of the silver. It is then dissolved in six solutions of diluted hydrochloric acid, unless it has been roasted in revolving furnaces, when nearly all the copper present is soluble in water. The process lasts two days, and each trough is reckoned to yield about 5 tons of dissolved pyrites per day. Of these six solutions the first is the original weak copper solution, which, after standing some time, loses part of its lime and sodium salts by evaporation. The others are formed with water from the condensing tower, impregnated with acid. The extent to which the different metals are dissolved in the solution varies according to the type of furnace used. If the pyrites be roasted in ordinary furnaces, about 1·94 per cent. of the copper is soluble in water, in rotary furnaces 2·77 per cent., the total proportion of copper being about 3 per cent.

Three products are thus obtained—two solutions and an insoluble residuum, containing 90 to 95 per cent. of ferric oxide, and utilized in ironworks under the name of purple ore or blue billy. The proportion of iron is from 63 to 66 per cent.; Spanish pyrites yields the best residuum. It is used to feed puddling-furnaces and for fusing metals.

Chloride of silver is found in the first solution, to which a little salt is sometimes added. The silver is usually separated by the Claudet method, but the process varies in different works. It is precipitated in the form of iodide of silver, by means of a soluble metal iodide, after the solution has become clear by standing. As the silver floats in the fluid in such minute particles that it takes a long time to settle, various means are used to hasten the process. These are generally kept secret, but mainly consist in the addition of sugar of lead, glue, tannic acid, or some other substance which settles quickly, and carries down with it the fine iodide of silver. The deposit thus obtained, containing iodides of lead and silver, and sulphate of lead, is variously treated with hydrochloric acid, zinc, sulphide of sodium, etc., the aim of all the methods being to separate the iodine so that it may be used again. A composite metal is produced, containing about 6 per cent. of silver, which is seldom worked on the spot, but is usually sent to the silver refineries at Hamburg and elsewhere. In the Gibb process, 6 per cent. of the copper is separated from the first solution by sulphuretted hydrogen, and almost all the silver is found in it. Most of the silver can also be extracted if 19 per cent. of the copper be separated from the solution with iron. Another method is to extract the copper and silver by electrolysis.

In precipitating the copper, if the water solutions obtained from the pyrites are treated alone, copper of the best quality is produced from them, and simply refined after smelting. Generally, however, the water and acid solutions are mixed, and the copper separated with scrap-iron in troughs, steam being used to precipitate it. The time required for the process varies; the quantity of iron is generally 1 part by weight to 1 of copper. The composition of the precipitated copper varies

according to the quality of the pyrites, the purity and quality of the iron used, and the strength of the solution. In Belgium it contains nearly 76 per cent. of pure copper, in Silesia 80 per cent., and small quantities of iron, silver, etc.

The copper obtained from the water solutions alone may, with the addition of chalk, slag, etc., be melted down into coarse copper in an ordinary furnace. If the copper be impure, sulphur must be added to it, and the mixture again roasted and smelted; while if it contains much carbon part must be oxidized by roasting, the carbon burnt, and the copper smelted afterwards with the same quantity of unburnt copper. Generally, however, the whole of the precipitated copper, whatever its quality, is smelted all together in ordinary refining-furnaces.

In smaller works blue vitriol, not coarse copper, is produced from the precipitated metal. The iron compounds are transformed, by roasting, into iron oxide. In Bohemia, the copper and silver from the first water solution are separated, and the precipitated copper is freed from the silver by the Philipp system. Salt, both raw and in a hot solution, is used to convert the copper into oxide, and the silver into chloride. Another and better method is to smelt the copper and separate the two metals electrolytically; copper of the finest quality is thus obtained, and all the silver settles as slag. After the copper has been deposited, the solution contains principally iron and sulphate of soda, and cannot at present be utilized, because of the expense of separating the iron. B. D.

RECENT ADVANCES IN PYROMETRY.

By W. C. ROBERTS-AUSTEN. *Transactions of the American Institute of Mining Engineers*, 1893, *vol. xxiii., pages* 407-443.

The earliest pyrometers were essentially thermoscopes, and although their graduation presented great difficulties, their importance for the measurement of high temperatures has been recognized for centuries.

The author gives a brief history of pyrometers from their commencement. In 1886, very distinct advances in practical pyrometry were made, and Mr. Callendar and Mr. Le Chatelier then brought out their pyrometers. Sir William Siemens' instrument was brought out in 1860, and was the first to use the electrical resistance of a heated metallic conductor for measuring high temperatures. The Le Chatelier instrument was the first to use a platinum-rhodium couple. Photography is used for recording the temperature employed in conducting metallurgical operations.

The calorimetric method of Mr. Violle gave very trustworthy results, which are used for graduating other forms of pyrometers.

The pyrometric alloys used by Mr. Prinsep in 1827 were too cumbersome and difficult for ordinary use.

The Le Chatelier photometric instrument and the optical pyrometer of Messrs. Nouel and Mesuré are figured and described.

It is an important point to know to what extent the indications of different pyrometers are concordant, and to this end Prof. Barns devised an apparatus for comparing the air-thermometer with the thermo-couple, and the results he obtained agreed remarkably well. Prof. Barns also invented a method depending on the viscosity of gases.

Of the various forms of pyrometers described, the author is in favour of the electrical ones, which involve the use of the galvanometer. A. W. G.

METRIC CONVERSION TABLES.

I.—Tables for Converting British to Metric Weights and Measures.

	Linear.				Capacity.			
	Inches to Millimetres.	Feet to Metres.	Yards to Metres.	Miles to Kilometres.	Fluid Drams to Millilitres or Cubic Centimetres.	Fluid Ounces to Millilitres.	Imperial Quarts to Litres.	Imperial Gallons to Litres.
1	25·3995	0·304794	0·914383	1·60931	3·54	28·39	1·13586	4·54345
2	50·7991	0·609589	1·828767	3·21863	7·09	56·79	2·27173	9·08691
3	76·1986	0·914383	2·743150	4·82794	10·64	85·19	3·41759	13·63037
4	101·5982	1·219178	3·657534	6·43726	14·19	113·58	4·54346	18·17383
5	126·9977	1·523972	4·571917	8·04657	17·74	141·98	5·67932	22·71728
6	152·3972	1·828767	5·486301	9·65589	21·29	170·38	6·81519	27·26075
7	177·7968	2·133561	6·400684	11·26520	24·84	198·77	7·95105	31·80421
8	203·1963	2·438356	7·315068	12·87452	28·39	227·17	9·08692	36·34766
9	228·5959	2·743150	8·229451	14·48383	31·94	255·57	10·22278	40·89112

	Square.				Cubic.				Pounds per Square Inch to Kilogrammes per Square Centimetre.
	Square Inches to Square Centimetres.	Square Feet to Square Decimetres.	Square Yards to Square Metres.	Acres to Hectares.	Cubic Inches to Cubic Centimetres.	Cubic Feet to Cubic Metres.	Cubic Yards to Cubic Metres.	Bushels to Hectolitres.	
1	6·451	9·290	0·836	0·4047	16·386	0·02832	0·765	0·36347	0·07030
2	12·903	18·580	1·672	0·8093	32·772	0·05663	1·529	0·72695	0·14061
3	19·354	27·870	2·508	1·2140	49·159	0·08495	2·293	1·09043	0·21092
4	25·805	37·160	3·344	1·6187	65·546	0·11326	3·058	1·47391	0·28123
5	32·257	46·450	4·180	2·0233	81·931	0·14158	3·823	1·81738	0·35154
6	38·708	55·740	5·017	2·4280	98·317	0·16989	4·587	2·18086	0·42185
7	45·160	65·030	5·853	2·8327	114·703	0·19821	5·352	2·54434	0·49215
8	51·611	74·320	6·689	3·2374	131·089	0·22652	6·116	2·90781	0·56246
9	58·062	83·610	7·525	3·6420	147·476	0·25484	6·881	3·27129	0·63187

	Weight.				
	Grains to Milligrammes.	Avoirdupois Ounces to Grammes.	Troy Ounces to Grammes.	Avoirdupois Pounds to Kilogrammes.	Avoirdupois Tons to Metric Tonnes.
1	64·7989	28·3495	31·10349	0·45359	1·01605
2	129·5979	56·6991	62·20699	0·90719	2·03210
3	194·3968	85·0486	93·31049	1·36078	3·04814
4	259·1958	113·3982	124·41398	1·81437	4·06419
5	323·9947	141·7477	155·51748	2·26796	5·08024
6	388·7937	170·0972	186·62098	2·72156	6·09628
7	453·5926	198·4468	217·72447	3·17515	7·11233
8	518·3916	226·7963	248·82797	3·62874	8·12838
9	583·1905	255·1458	279·93146	4·08233	9·14443

Miscellaneous.

1 inch	= 25·39954113188991 millimetres.
1 foot	= 0·304794493582 metre.
1 yard	= 0·914383480748 metre.
1 fathom	= 1·829 metres.
1 chain	= 20·116436576456 metres.
1 mile	= 1·6093149261 kilometres.
1 square inch	= 6·451366897 square centimetres.
1 square foot	= 0·092899683 square metre.
1 square yard	= 0·836097149 square metre.
1 square mile	= 259 hectares.
1 avoir. pound	= 453·592652546 grammes.
1 gallon	= 4·54345797 litres.
1 bushel	= 36·34766376 litres.

II.—Tables for Converting Metric Weights and Measures to British.

	Linear.				Capacity.				
	Metres to Inches.	Metres to Feet.	Metres to Yards.	Kilometres to Miles.	Millilitres or Cubic Centimetres to Fluid Drams.	Centilitres to Fluid Ounces.	Litres to Imperial Quarts.	Decalitres to Imperial Gallons.	Hectolitres to Imperial Bushels.
1	39·37	3·28090	1·093633	0·62138	0·28	0·352	0·8804	2·2010	2·7512
2	78·74	6·56180	2·187266	1·24276	0·56	0·704	1·7608	4·4019	5·5024
3	118·11	9·84270	3·280899	1·86415	0·84	1·056	2·6412	6·6029	8·2536
4	157·48	13·12360	4·374532	2·48553	1·13	1·409	3·5215	8·8039	11·0048
5	196·85	16·40450	5·468165	3·10691	1·41	1·761	4·4019	11·0048	13·7560
6	236·22	19·68540	6·561798	3·72829	1·69	2·113	5·2823	13·2058	16·5073
7	275·59	22·96629	7·655431	4·34968	1·97	2·465	6·1627	15·4068	19·2585
8	314·97	26·24719	8·749064	4·97106	2·25	2·817	7·0431	17·6077	22·0097
9	354·34	29·52809	9·842697	5·59244	2·53	3·169	7·9235	19·8087	24·7609

	Square.				Cubic.				
	Square Centimetres to Square Inches.	Square Metres to Square Feet.	Square Metres to Square Yards.	Hectares to Acres.	Cubic Centimetres to Cubic Inches.	Cubic Decimetres to Cubic Inches.	Cubic Metres to Cubic Feet.	Cubic Metres to Cubic Yards.	Cubic Metres to Imperial Gallons.
1	0·155	10·764	1·196	2·471	0·0610	61·027	35·316	1·308	220·097
2	0·310	21·529	2·392	4·942	0·1220	122·054	70·633	2·616	440·193
3	0·465	32·293	3·588	7·413	0·1831	183·081	105·950	3·924	660·290
4	0·620	43·057	4·784	9·885	0·2441	244·108	141·266	5·232	880·387
5	0·775	53·821	5·980	12·356	0·3051	305·135	176·583	6·540	1100·483
6	0·930	64·586	7·176	14·827	0·3662	366·162	211·899	7·848	1320·580
7	1·085	75·350	8·372	17·298	0·4272	427·189	247·216	9·156	1540·677
8	1·240	86·114	9·568	19·769	0·4882	488·216	282·533	10·464	1760·773
9	1·395	96·879	10·764	22·240	0·5492	549·243	317·849	11·772	1980·870

	Weight.							
	Milli-grammes to Grains.	Kilogrammes to Grains.	Hecto-grammes to Avoir-dupois Ounces.	Kilogrammes to Avoirdu-pois Pounds.	Quintals to Avoir-dupois Pounds.	Tonnes or Milliers to Avoir-dupois Pounds.	Grammes to Troy Ounces.	Tonnes to Tons.
1	0·01543	15432·35	3·5274	2·20462	220·46	2204·6	0·03215	0·9842
2	0·03086	30864·70	7·0548	4·40924	440·92	4409·2	0·06430	1·9684
3	0·04630	46297·04	10·5822	6·61386	661·39	6613·9	0·09645	2·9526
4	0·06173	61729·39	14·1096	8·81848	881·85	8818·5	0·12860	3·9368
5	0·07716	77161·74	17·6370	11·02311	1102·31	11023·1	0·16075	4·9210
6	0·09259	92594·09	21·1644	13·22773	1322·77	13227·7	0·19290	5·9052
7	0·10803	108026·44	24·6918	15·43235	1543·24	15432·3	0·22505	6·8894
8	0·12346	123458·79	28·2192	17·63697	1763·70	17637·0	0·25721	7·8736
9	0·13889	138891·14	31·7465	19·84159	1984·16	19841·6	0·28936	8·8579

Miscellaneous.

1 millimetre	=	0·03937079 inch.
1 metre	=	39·37079 inches.
	=	3·28089916 feet.
	=	1·09363305 yards.
1 kilometre	=	0·62138241792 mile.
1 square centimetre	=	0·1550059105 square inch.
1 square metre	=	10·76429934 square feet.
	=	1·19603326 square yards.
1 hectare	=	2·4711430996 acres.
1 cubic centimetre	=	0·061027 cubic inch.
1 cubic metre	=	35·31658074 cubic feet.
1 litre	=	61·0270515 cubic inches.
	=	0·22009668 gallons.
1 kilogramme	=	15,432·34874 grains.
1 tonne or millier	=	0·984205914 tons.

M. W. B.

LIGNUM VITÆ FOR AXLE-BOXES.

Pockholz im Maschinenbau. Anon. *Dingler's Polytechnisches Journal*, 1893, *vol. ccxc.*, *page* 288.

Lignum vitæ has been found, by direct experiment at the Conservatoire des Arts et Métiers at Paris, to withstand about one and a half times as much pressure as oak. Experiments on its frictional resistance have also been made. Two axle-boxes were taken, one fitted with ordinary brasses, and the other with lignum vitæ bearings. The frictional resistance of the first being taken as 1, that of the second was (1) 0·661, (2) 0·748, and (3) 0·785, each being loaded with (1) 220·4 lbs., (2) 330·6 lbs., and (3) 440·8 lbs. respectively. The ordinary bearings were also found to be heated, while the wooden ones remained cool. In spite of this low coefficient of friction the author finds that lignum vitæ is also good for brake blocks.

W. M.

THE HANARTE VENTILATOR.

Le Ventilateur Hanarte. By G. HANARTE. *Publications de la Société des Ingénieurs sortis de l'Ecole Provinciale d'Industrie et des Mines du Hainaut*, 1893, *vol. iii., pages* 4-7.

A Hanarte ventilator has been erected at the Ste. Julie pit of the Vingt-quatre-actions collieries. The following table contains the results of experiments:—

	Hanarte.	
	No. 1.	No. 2
Diameter of ventilator, feet	9·84	13·12
Width of ventilator, inches	13·39	13·39
Volume or contents of ventilator, cubic feet	85	85
Revolutions per minute, engine	43	50
,, ,, ventilator	129	150
Volume per revolution, cubic feet	119	130
Ratio of volume of fan to volume of air produced per revolution	1·40	1·53
Water-gauge, inches	1·575	1·968
Volume of air per minute, cubic feet ...	15,370	19,460
Area of place of measurement, square feet...	27·3	27·3
Velocity at place of measurement, per minute, feet	563	713
Dimensions of engine, inches	12·6 × 20·5	12·6 × 20·5

When air was freely admitted from the atmosphere, the volume produced by the fan was 36,800 cubic feet per minute under a water-gauge of 0·591 inch, at 44 revolutions of the engine and 132 of the ventilator per minute, and exhausted 3·29 times its volume per revolution. M. W. B.

THE MEASUREMENT OF WIRE ROPES.

By A. S. HALLIDIE. *Engineering and Mining Journal* (*New York*), 1892, *vol. liii., page* 426.

A custom has obtained of measuring ropes by their diameters, and as a consequence many disputes have arisen between the seller and buyer regarding the actual size of such ropes. The method that has generally prevailed, heretofore, has been to measure ropes by their circumference, and this custom is still in use where ordinary technical or business accuracy is required, and no other way of measuring a rope can be relied on for accuracy and fairness. This applies equally to all kinds of ropes, whether three, four, or six-stranded.

Take for example a rope made of six strands; there being no accepted rule as to where the diameter shall be measured, it may be taken at any part of the circumference of the hexagon formed by the six strands. Hence it is much more satisfactory to measure the circumference of the rope; but here again there is a possibility of unfairness, in the absence of a defined method of measurement. The best way, according to the author, is to take a piece of ordinary writing paper, about ¼ inch wide, wrap it round the rope, and cut it off with a sharp knife at the overlap. This will give the circumference measured on the faces of the hexagon, and determine exactly the size of the rope.

Each division of space of the circumference being 3·1416 times that of the diameter, the circumference measurement is three times as accurate, and in every way, both as to fairness, accuracy and simplicity, this method is to be preferred.

J. W.

II.—BAROMETER, THERMOMETER, ETC., READINGS FOR THE YEAR 1894.

BY M. WALTON BROWN.

The barometer, thermometer, etc., readings have been supplied by permission of the authorities of Glasgow and Kew Observatories, and give some idea of the variations of atmospheric temperature and pressure in the intervening districts in which mining operations are chiefly carried on in this country.

The barometer at Kew is 34 feet, and at Glasgow is 180 feet, above sea-level. The barometer readings at Glasgow have been reduced to 32 feet above sea-level, by the addition of 0·150 inch to each reading, and the barometrical readings at both observatories are reduced to 32 degs. Fahr.

The statistics of fatal explosions in collieries are obtained from the annual reports of H.M. Inspectors of Mines, and are also printed upon the diagrams (Plates XXVIII. and XXIX.) recording the meteorological observations.

DEATHS FROM EXPLOSIONS OF FIRE-DAMP DURING 1894, IN THE SEVERAL INSPECTION DISTRICTS.

	Deaths.
Durham	2
Liverpool	0
Manchester	1
Midland	0
Newcastle-upon-Tyne	1
Scotland, East	1
Do. West	5
South Wales	295
South-Western	0
Staffordshire, North	1
Do. South	8
Yorkshire	3
Total	317

List of Fatal Explosions in Collieries during 1894.

1894.	Colliery.	District.	Deaths.
Jan. 1, 1·30 a.m.	Gorscinon	South Wales	1
Mar. 23, 3· 0 p.m.	Ayr-Sundrum (No. 3)	Scotland, West ...	1
April 18, 6· 0 „	Portland (No. 5) ...	Do. do. ...	1
May 19, 12· 0 „	Allerton Main... ...	Yorkshire	3
June 23, 3·50 „	Albion	South Wales	290
Aug. 1, 12·30 „	Llwynpia (No. 3) ...	Do.	1
„ 8, 5· 0 „	Bank House	Manchester	1
„ 9, 5· 0 „	Rowley Hall	Staffordshire, South ...	1
„ 17, 6·30 „	Black Boy	Durham	1
Sept. 6, 2·30 „	West Auckland ...	Do.	1
„ 10, 7·30 a.m.	Radcliffe	Newcastle-upon-Tyne	1
„ 21, 12·15 p.m.	East Cannock ...	Staffordshire, South ...	1
„ 26, 2·30 „	Wednesbury	Do. do.	1
„ 27, 9· 0 a.m.	International Anthracite	South Wales	1
Oct. 15, 4· 0 p.m.	Common (No. 10) ...	Scotland, West ...	1
„ 16, 5·30 „	Harecastle and Woodshutts colliery, Moss pits	Staffordshire, North ...	1
„ 17, 7·15 a.m.	Ellismuir (No. 2) ...	Scotland, West ...	1
„ 25, 11· 0 p.m.	Blairhall	Scotland, East ...	1
„ 27, 8· 0 a.m.	Common (No. 14) ...	Scotland, West ...	1
„ 27, 7·30 p.m.	Sandwell Park ...	Staffordshire, South ...	4
Nov. 10, 9·30 a.m.	Gadlys, Dyllas ...	South Wales	2
„ 22, 2·30 p.m.	Ashmore Park... ..	Staffordshire, South ...	1
		Total ...	317

JANUARY, 1894.

KEW.								GLASGOW.							
	Barometer.				Temperature.				Barometer.				Temperature.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	30·307	30·304	30·242	30·239	37·9	32·4	N	1	30·430	30·444	30·403	30·395	41·6	29·5	W
2	30·182	30·182	30·282	30·396	36·9	26·3	NNE	2	30·369	30·386	30·459	30·577	36·3	27·8	ENE
3	30·457	30·543	30·511	30·436	30·2	26·7	ENE	3	30·615	30·676	30·671	30·658	34·6	31·1	E
4	30·336	30·224	30·060	30·006	29·0	16·9	NE	4	30·598	30·509	30·382	30·265	34·3	31·2	NE
5	29·827	29·820	29·681	29·677	22·2	14·0	ESE	5	30·145	29·987	29·779	29·626	33·0	28·9	E
6	29·626	29 612	29·636	29·713	26·5	21·6	NNE	6	29·479	29 584	29·592	29·638	30·1	16·7	E
7	29·777	29·893	29·950	30·041	25·1	20·1	E	7	29·669	29·722	29·770	29·875	33·8	15·6	ENE
8	30·069	30·038	29·912	29·895	36·8	30·7	E	8	29·914	29·919	29·817	29·707	34·3	25·7	E
9	29·797	29·784	29·753	29·816	38·5	31·7	SE	9	29·592	29·621	29·642	29·656	40·3	31·1	NE
10	29·825	29·834	29·845	29·836	48·1	30·0	S	10	29·468	29·378	29·518	29·467	49·1	35·8	S
11	29 823	29·875	29·865	29·803	51·3	39·9	SSW	11	29·399	29·346	29·578	29·632	50·9	44·6	S
12	29·768	29·963	30·083	30·114	50·2	41·6	SW	12	29·480	29·584	29·731	29·768	47·2	41·0	SSW
13	30·046	30·000	29·933	29·863	47·9	42·8	S	13	29·570	29·596	29·604	29·594	49·3	44·3	SSW
14	29·798	29·758	29·753	29·843	44·6	42·2	E	14	29·626	29·701	29·736	29·799	46·6	35·6	NW
15	29·941	30·059	29·983	29·871	46·3	37·9	SW	15	29·806	29·809	29·624	29·475	43·1	35·8	SE
16	29·840	29·879	29·824	29·797	49·9	44·0	SW	16	29·468	29·428	29·398	29·436	47·8	38·4	SW
17	29·695	29·619	29·464	29·446	51·1	44·2	SW	17	29·246	29·187	29·107	29·098	46·3	42·3	SW
18	29·400	29·502	29·604	29·753	49·0	41·0	W	18	29·166	29·278	29·382	29·554	45·1	38·9	WSW
19	29·864	29·931	29·835	29·694	48·9	37·3	SW	19	29·615	29·554	29·308	29·084	47·5	39·1	SSW
20	29·546	29·564	29·526	29·722	49·8	42·9	SW	20	28·908	28·977	29·192	29·417	45·8	41·0	WSW
21	29·875	29·951	29·853	29·808	48·1	40·1	SW	21	29·490	29·468	29·352	29·327	46·2	38·2	SW
22	29·669	29·573	29·401	29·478	47·1	37·8	SW	22	29·251	29·210	29·144	29·421	40·8	35·1	SW
23	29·575	29·812	29·931	30·107	39·9	30·6	N	23	29·603	29·814	29·929	29·946	35·4	29·0	WNW
24	30·176	30·193	30·056	29·964	44·9	26·2	SSW	24	29·790	29·622	29·470	29·447	47·9	32·0	S
25	29·905	29·875	29·726	29·528	48·2	44·9	SSW	25	29 522	29·502	29·310	29·278	47·1	35·1	WSW
26	29·442	29·698	29·846	29·880	45·4	37·8	W	26	29·335	29·407	29·407	29·234	41·9	31·1	SW
27	29·803	29·773	29·635	29·542	51·2	43·9	SW	27	29·142	29·030	29·021	29·060	47·9	35·2	WSW
28	29·580	29·693	29·676	29·788	46·8	35·8	W	28	29·080	29·044	29·276	29·619	38·4	32·6	SW
29	29·937	30·067	30·012	29·788	43·8	34·9	W	29	29·766	29·779	29·424	29·052	44·7	31·9	S
30	29·643	29·740	29·728	29·567	47·5	40·1	W	30	29·082	29·159	29·138	29·128	44·8	32·2	SW
31	29·339	29·288	29·394	29·433	45·1	34·7	WNW	31	29·102	29·145	29·231	29·409	36·0	31·6	SW

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1	29·695	29·882	29·896	29·827	49·2	33·4	W	1	29·495	29·562	29·338	29·424	45·3	29·3	SSW
2	29·936	30·043	30·016	30·010	52·6	47·4	SW	2	29·547	29·538	29·438	29·492	49·6	44·7	SW
3	29·961	30·019	30·196	30·381	50·5	38·6	WNW	3	29·548	29·729	29·985	30·115	47·0	38·8	WSW
4	30·437	30·456	30·382	30·370	48·9	34·8	SW	4	30·105	30·046	29·920	30·051	49·5	39·7	SSW
5	30·368	30·391	30·363	30·364	49·2	41·1	WSW	5	30·159	30·238	30·201	30·069	44·3	36·9	SSW
6	30·297	30·231	30·094	30·003	51·7	41·0	SW	6	29·772	29·530	29·362	29·322	50·9	39·0	SW
7	29·934	29·936	29·890	29·913	55·0	50·4	SW	7	29·279	29·327	29·250	29·509	52·4	39·6	SW
8	30·012	30·230	30·272	30·252	53·0	40·7	WNW	8	29·676	29·867	29·862	29·809	45·3	39·4	WSW
9	30·095	29·880	29·807	29·851	52·3	39·8	SW	9	29·588	29·444	29·424	29·344	45·3	38·5	SW
10	29·797	29·768	29·689	29·640	49·4	42·0	WSW	10	29·236	29·253	29·268	29·255	40·7	35·1	W
11	29·524	29·564	29·483	29·288	53·1	46·4	WSW	11	29·198	28·996	28·738	28·501	48·1	35·3	NE
12	29·339	29·546	29·615	29·675	49 8	37·7	W	12	28·604	29·122	29·278	29·439	43·5	32·7	W
13	29·737	29·978	30·071	30·105	44·1	34·1	NW	13	29·699	29·867	29·907	30·024	39·8	31·1	W
14	30·109	30·157	30·156	30·167	42·1	29·9	W	14	30·072	30·110	30·082	30·068	34·2	27·1	NE
15	30·145	30·139	30·087	30·096	48·1	36·9	W	15	30·020	29·997	29·950	29·931	35·6	30·7	ENE
16	30·043	30·047	30·040	30·087	47·3	39·2	WSW	16	29·885	29·890	29·883	29·914	41·6	35·0	ENE
17	30·111	30·131	30·046	30·039	41·2	37·9	ESE	17	29·938	30·004	30·074	30·186	40·9	34·8	NE
18	30·149	30·240	30·423	30·526	37·9	29·1	E	18	30·251	30·342	30·358	30·391	38·5	34·0	SSE
19	30·515	30·523	30·479	30·471	37·4	27·0	E	19	30·362	30·362	30·309	30·304	40·4	31·4	S
20	30·431	30·447	30·374	30·377	37·5	25·8	E	20	30·311	30·372	30·332	30·290	41·1	36·9	SSW
21	30·325	30·309	30·249	30·298	36·9	23·0	E	21	30·192	30·185	30·171	30·245	44·4	38·7	WSW
22	30·299	30·320	30·218	30·179	39·2	23·2	NE	22	30·271	30·254	30·079	29·891	43·4	33·9	WNW
23	30·080	30·006	29·821	29·657	44·0	28·0	SW	23	29·740	29·598	29·042	29·202	47·0	38·7	SSW
24	29·539	29·660	29·795	29·890	47·6	34·2	W	24	29·254	29·364	29·476	29·576	39·7	32·7	W
25	29·889	29·762	29·616	29·589	52·2	34·2	SW	25	29·458	29·097	29·069	29·110	49·0	33·9	SW
26	29·684	29·714	29·731	29·803	52·9	48·4	SW	26	29·264	29·273	29·341	29·194	49·4	37·1	SW
27	29·799	29·827	29·911	30·033	54·2	39·1	W	27	29·284	29·448	29·592	29·688	49·7	35·6	SW
28	29·961	29·873	29·901	30·072	47·3	36·5	W	28	29·525	29·569	29·642	29·618	41·9	35·5	SW

MARCH, 1894.

KEW.								GLASGOW.							
	Barometer.				Temperature.				Barometer.				Temperature.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	29·995	29·906	29·805	29·791	48·7	37·6	SW	1	29·258	29·253	29·274	29·290	45·2	39·3	SW
2	29·925	30·063	30·094	30·197	50·5	36·9	W	2	29·334	29·555	29·780	29·910	45·8	36·2	WSW
3	30·194	30·201	30·116	30·084	49·1	29·5	SW	3	29·910	29·860	29·815	29·774	44·4	36·1	SW
4	30·020	30·037	29·987	30·149	50·2	34·6	W	4	29·606	29·678	29·955	30·112	45·3	35·4	NW
5	30·299	30·327	30·168	29·940	47·6	32·4	WSW	5	30·089	29·956	29·538	29·374	47·6	35·0	SSW
6	29·665	29·618	29·596	29·684	50·5	41·8	WNW	6	29·211	29·298	29·515	29·688	45·7	38·6	WNW
7	29·795	29·858	29·827	29·752	47·8	35·6	WSW	7	29·719	29·695	29·546	29·320	42·6	32·7	SSE
8	29·652	29·603	29·575	29·658	48·6	39·3	SW	8	29·119	29·315	29·332	29·331	45·5	37·4	SW
9	29·650	29·616	29·547	29·532	49·5	39·0	SW	9	29·242	29·155	29·059	29·194	48·4	38·7	SSW
10	29·572	29·670	29·692	29·711	53·2	44·3	WSW	10	29·235	29·312	29·328	29·125	43·4	34·6	SW
11	29·558	29·473	29·538	29·649	54·0	40·2	W	11	28·756	28·993	29·059	29·152	48·1	34·3	SW
12	29·640	29·667	29·601	29·411	50·9	39·9	W	12	29·156	29·175	29·160	29·096	42·6	34·0	SW
13	29·030	29·225	29·341	29·376	49·0	39·0	W	13	28·920	28·870	28·981	29·141	45·0	35·1	W
14	29·457	29·539	29·456	29·359	48·8	35·9	W	14	29·246	29·306	29·279	29·322	44·1	34·3	SW
15	29·246	29·411	29·528	29·673	48·9	37·0	N	15	29·364	29·442	29·510	29·630	43·6	31·9	WNW
16	29·742	29·828	29·876	30·014	47·7	32·5	N	16	29·716	29·835	29·905	30·008	45·1	30·7	W
17	30·097	30·206	30·200	30·267	46·4	29·8	N	17	30·064	30·122	30·102	30·133	47·7	31·3	W
18	30·265	30·308	30·246	30·293	51·5	30·7	NE	18	30·117	30·159	30·149	30·184	48·0	39·7	SW
19	30·286	30·319	30·259	30·271	51·9	29·1	E	19	30·174	30·230	30·220	30·254	50·4	45·1	W
20	30·244	30·258	30·221	30·230	53·2	42·9	NE	20	30·223	30·246	30·208	30·208	47·2	44·1	SW
21	30·199	30·236	30·201	30·272	52·9	39·9	E	21	30·171	30·206	30·190	30·254	53·3	42·1	SW
22	30·288	30·348	30·345	30·402	46·1	40·1	E	22	30·268	30·315	30·294	30·362	56·1	37·9	NE
23	30·409	30·458	30·397	30·410	52·4	40·1	ENE	23	30·394	30·448	30·416	30·444	58·7	35·9	NW
24	30·376	30·357	30·249	30·219	55·4	37·2	E	24	30·432	30·401	30·281	30·270	59·1	37·7	ESE
25	30·151	30·142	30·066	30·048	56·5	39·7	E	25	30·209	30·169	30·064	30·062	58·8	37·3	ENE
26	30·011	30·020	29·971	30·030	59·3	33·8	E	26	30·035	30·019	29·946	29·996	57·6	37·2	E
27	30·025	30·092	30·061	30·165	62·0	29·5	E	27	30·041	30·111	30·098	30·200	54·7	36·3	E
28	30·191	30·243	30·206	30·250	55·7	29·8	E	28	30·222	30·242	30·158	30·151	52·6	37·5	NE
29	30·235	30·229	30·105	30·100	58·1	40·3	E	29	30·102	30·074	29·992	29·974	60·1	35·8	SW
30	29·963	29·884	29·753	29·757	62·7	35·0	SE	30	29·894	29·826	29·696	29·665	60·7	37·0	SSW
31	29·700	29·741	29·777	29·890	63·0	45·1	SW	31	29·643	29·696	29·728	29·842	57·7	41·8	SSE

APRIL, 1894.

1	29·943	30·010	29·964	29·931	65·2	41·2	SW	1	29·891	29·968	29·948	29·959	56·1	41·9	S
2	29·935	29·907	29·831	29·835	63·0	39·0	ESE	2	29·915	29·891	29·833	29·846	55·2	38·7	E
3	29·815	29·829	29·802	29·894	65·1	37·4	SSE	3	29·842	29·879	29·874	29·941	48·4	40·9	E
4	29·961	30·060	30·064	30·134	58·9	41·3	ENE	4	29·983	30·089	30·119	30·203	47·4	39·8	E
5	30·131	30·127	30·045	30·063	58·9	39·5	ENE	5	30·208	30·242	30·197	30·234	52·9	40·8	NE
6	29·981	29·967	29·945	29·974	54·0	44·3	E	6	30·210	30·165	30·070	30·101	48·8	39·7	ENE
7	29·946	29·955	29·872	29·901	64·1	43·3	ENE	7	30·062	30·054	30·008	29·996	48·7	41·4	NE
8	29·885	29·888	29·821	29·871	70·2	46·1	S	8	29·925	29·906	29·846	29·833	48·9	40·5	E
9	29·910	29·980	29·984	30·036	63·2	44·1	SW	9	29·772	29·755	29·769	29·841	53·1	40·8	SSW
10	30·044	30·050	29·984	29·994	68·6	37·1	S	10	29·869	29·903	29·892	29·899	57·3	46·6	S
11	29·930	29·879	29·767	29·779	70·0	46·5	S	11	29·894	29·915	29·842	29·834	61·9	49·6	ENE
12	29·717	29·759	29·742	29·778	60·0	45·9	SW	12	29·788	29·767	29·752	29·840	56·8	43·7	ENE
13	29·766	29·784	29·748	29·775	59·3	42·6	WNW	13	29·732	29·834	29 790	29·770	46·4	41·1	ESE
14	29·704	29·703	29·694	29·732	54·1	41·7	S	14	29·798	29·614	29·510	29·515	50·1	35·6	ESE
15	29·746	29·730	29·666	29·593	60·1	48·6	S	15	29·622	29·616	29·520	29·530	56·2	41·9	SE
16	29·436	29·362	29·383	29·438	54 9	47·4	S	16	29·499	29·437	29·346	29·306	55·1	39·9	NE
17	29·466	29·489	29·483	29·536	57·9	43·9	S	17	29·277	29·351	29·422	29·523	50·7	44·4	NNE
18	29·553	29·645	29·701	29·831	60·6	44·4	ESE	18	29·593	29·736	29·834	29·984	54·5	44 9	N
19	29·889	30·013	30·060	30·133	49·1	43·5	NNE	19	30·044	30·102	30·096	30·135	56·9	43·4	NW
20	30·127	30·137	30·092	30·089	49·2	42·8	NNE	20	30·120	30·098	30·018	30·020	60·3	36·8	S
21	30·062	30·063	29·979	30·005	53·2	38·8	NE	21	30·002	29·980	29·919	29·934	59·9	38·6	S
22	29·948	29·909	29·771	29·714	53·5	33·5	E	22	29·880	29·839	29·729	29·719	55·8	39·3	E
23	29·602	29·585	29·584	29·626	56·9	40·9	ESE	23	29·647	29·610	29·575	29·566	47·6	42·1	E
24	29·645	29·668	29·673	29·705	57·0	43·3	S	24	29·508	29·448	29·399	29·441	53·8	42·0	SE
25	29·663	29·654	29·695	29·718	58·3	44·8	WSW	25	29·446	29·441	29·430	29·446	54·9	42·2	SSE
26	29·696	29·685	29·643	29·615	59·3	46·4	S	26	29·480	29·502	29·483	29·511	55·8	44·6	S
27	29·560	29·643	29·646	29·689	59·1	45·9	SW	27	29·521	29·549	29·602	29·709	54·4	43·5	E
28	29·738	29·860	29·963	30·092	56·1	42·6	N	28	29·741	29·955	29·965	30·000	60·9	41·7	N
29	30·106	30·107	30·015	30·014	63·0	37·9	NW	29	29·960	29·956	29·946	29·986	57·6	47·2	SW
30	29·985	30·035	30·085	30·231	57·0	44·5	W	30	30·073	30·208	30·252	30·363	58·3	46·8	WNW

MAY, 1894.

KEW.								GLASGOW.							
	BAROMETER.				TEMPERATURE.				BAROMETER.				TEMPERATURE.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	30·242	30·288	30·260	30·293	54·0	40·2	N	1	30·372	30·352	30·288	30·242	57·1	40·1	W
2	30·238	30·197	30·085	30·005	53·6	35·8	WNW	2	30·110	29·979	29·827	29·690	51·3	43·9	SW
3	29·878	29·822	29·774	29·790	59·9	47·7	NW	3	29·669	29·672	29·622	29·674	51·2	41·7	WNW
4	29·776	29·803	29·850	29·918	53·6	42·4	NW	4	29·662	29·738	29·806	29·844	48·9	36·0	WNW
5	29·906	29·921	29·833	29·820	58·9	39·6	W	5	29·816	29·788	29·688	29·545	50·6	37·2	WSW
6	29·775	29·758	29·738	29·788	58·5	46·7	SW	6	29·509	29·511	29·539	29·562	51·3	42·5	W
7	29·785	29·835	29·845	29·951	60·7	41·9	WNW	7	29·526	29·564	29·632	29·734	50·8	40·3	WSW
8	29·974	30·012	29·985	29·966	56·6	44·9	SW	8	29·728	29·684	29·631	29·563	53·1	44·0	SW
9	29·884	29·802	29·703	29·766	56·9	44·1	SW	9	29·605	29·601	29·519	29·486	54·3	43·1	SW
10	29·782	29·761	29·759	29·805	55·4	41·9	SW	10	29·411	29·390	29·438	29·530	52·6	44·6	SSW
11	29·826	29·833	29·747	29·704	58·2	40·4	SSW	11	29·592	29·656	29·708	29·804	55·7	44·0	SW
12	29·776	29·943	30·010	30·090	57·2	45·1	N	12	29·886	29·962	30·029	30·090	54·4	39·1	WNW
13	30·084	30·080	30·021	30·003	60·1	41·1	NW	13	30·072	30·027	29·947	29·917	55·6	38·7	E
14	29·966	29·960	29·906	29·898	64·5	49·0	S	14	29·844	29·854	29·885	29·909	50·3	42·9	E
15	29·858	29·845	29·817	29·854	65·6	49·0	E	15	29·946	30·018	30·006	30·068	53·3	46·4	ENE
16	29·830	29·881	29·912	30·032	66·1	53·6	E	16	30·072	30·147	30·215	30·289	48·4	44·0	E
17	30·073	30·100	30·052	30·122	65·3	49·0	N	17	30·292	30·343	30·324	30·359	51·4	41·1	E
18	30·118	30·135	30·101	30·174	64·1	44·2	NE	18	30·321	30·319	30·326	30·381	54·1	41·7	E
19	30·174	30·184	30·097	30·061	50·7	41·8	NE	19	30·377	30·344	30·243	30·180	46·1	40·1	NE
20	29·970	29·939	29·897	29·929	48·7	36·8	N	20	30·105	30·038	29·978	29 996	49·2	32·1	N
21	29·919	29·922	29·904	29·968	52·2	33·8	N	21	29·988	30 001	29·974	30·041	50·6	32·4	NE
22	29·988	30·014	30·066	30·129	51·2	34·2	N	22	30·065	30·099	30·129	30·222	53·9	33·1	SSW
23	30·028	30·132	30·186	30·258	55·2	43·0	NE	23	30·297	30·359	30·378	30·437	58·5	36·3	E
24	30·258	30·246	30·184	30·210	63·9	45·0	NE	24	30·446	30·426	30·327	30·300	62·1	37·5	ENE
25	30·134	30·032	29·915	29·847	65·1	42 5	NNE	25	30·228	30·138	29·995	30·015	60·4	42·8	NW
26	29·803	29·760	29·684	29·681	52·1	42·5	NNW	26	29·963	29·926	29·946	30·005	54·8	42·1	N
27	29·601	29·649	29·676	29·691	51·4	41·9	N	27	29·940	29·858	29·735	29·678	54·3	38·4	NW
28	29·627	29·572	29·526	29·531	54·0	40·4	SSW	28	29·594	29·584	29·547	29·561	51·3	42·9	NW
29	29·537	29·588	29·602	29·669	55·4	40·0	WSW	29	29·561	29·611	29·616	29·685	56·8	42·8	NE
30	29·690	29·713	29·701	29·732	58·2	40·8	SW	30	29·666	29·661	29·609	29·633	53·9	40·0	E
31	29·735	29·747	29·769	29·844	58·7	40·7	WSW	31	29·626	29·654	29·676	29 733	55·1	41·2	SW

JUNE, 1894.

1	29·864	29·876	29·828	29·793	60·0	49·2	S	1	29·764	29·796	29·782	29·784	61·8	39·0	W
2	29·722	29·685	29·734	29·850	62·9	52·1	SW	2	29·718	29·681	29·694	29·803	58·0	44·9	ENE
3	29·902	29·972	29·951	29·954	64·4	50·5	S	3	29·871	29·946	29·950	29·982	54·2	47·7	E
4	29·867	29·756	29·699	29·782	66·0	52·9	ESE	4	29·938	29·898	29·838	29·786	53·1	45·7	E
5	29·784	29·791	29·735	29·729	63 2	52·5	SW	5	29·835	29·766	29·804	29·868	51·9	44·0	E
6	29·727	29·772	29·660	29·658	56·0	49·6	NE	6	29·898	29·897	29·826	29·829	59·3	39·1	SE
7	29·681	29·767	29·818	29·898	61·2	48·2	N	7	29·818	29·820	29·810	29·851	60·9	44·9	W
8	29·929	29·974	29·936	29·916	63·6	48·0	W	8	29·859	29·868	29·833	29·818	61·3	47·3	WSW
9	29·839	29·841	29·839	29·890	63 6	53·8	SW	9	29·784	29·823	29·848	29·877	56·8	50·2	ENE
10	29·886	29·850	29·742	29·688	62·3	52·4	WSW	10	29·806	29·678	29·578	29·60.	55·4	49·2	SSW
11	29·717	29·731	29·708	29·705	59·8	48·1	WNW	11	29·627	29·624	29·612	29·704	59·6	45·3	WNW
12	29·755	29·881	29·913	29·888	60·2	45·1	NW	12	29·811	29·855	29·846	29·906	65·3	49·1	W
13	29·843	29·948	29·997	30·083	56·7	49·9	N	13	29·940	29·987	29·994	30·042	68·3	43·8	NW
14	30·095	30·126	30·082	30·120	70·5	48·0	NW	14	30·064	30·070	30·049	30·019	61·1	44·1	W
15	30·110	30·107	30·081	30·060	67·0	51·3	W	15	29·932	29·857	29·914	30·028	60·8	51·6	SW
16	30·080	30·112	30·075	30·064	67·7	53·7	NW	16	30·086	30·057	29·954	29·883	64·1	43·6	SW
17	29·997	29·955	29·876	29·856	64·9	50·2	SW	17	29·718	29·611	29·666	29·686	58·3	49·0	WSW
18	29·827	29·768	29·704	29·844	61·1	50·3	SW	18	29·661	29·655	29·754	29 860	58·7	44·2	SW
19	29·995	30·083	30·067	30·066	65·6	47·9	W	19	29·925	29·916	29·859	29·789	58·1	44·5	SW
20	30·000	29·983	29·985	30·036	63·3	54·6	SW	20	29·794	29·885	29·946	30·019	60·3	50·9	WNW
21	30·096	30·162	30·136	30·169	72·5	51·3	SSW	21	30·055	30·066	30·044	30 045	64·0	42·3	WSW
22	30·145	30·125	30·073	30·070	69·6	51·2	SW	22	29·982	29·889	29·873	29·936	62·0	51·9	SSW
23	30·031	29·980	29·909	29·876	68·1	58·4	SW	23	29·885	29·774	29·594	29·604	58·0	50·6	SW
24	29·861	29·938	30·027	30·136	64·6	54·7	WNW	24	29·686	29·794	29·885	29·967	57·1	47·3	W
25	30·154	30·179	30·161	30·201	72·6	50·2	WSW	25	29·961	30·025	30·103	30·184	56·6	49·6	ENE
26	30·225	30·248	30·259	30·268	69·8	56·3	NNE	26	30·245	30·282	30·296	30·317	68·1	54·1	W
27	30·260	30·264	30·199	30·227	69·9	53·0	NE	27	30·322	30·342	30·307	30·350	71·1	55·3	E
28	30·232	30·247	30·203	30·269	76·5	52·0	E	28	30·314	30·333	30·318	30·343	72·0	52 0	ENE
29	30·309	30·317	30·297	30·340	77·1	51·2	NE	29	30·374	30·384	30·341	30·376	75·2	55·4	E
30	30·349	30·350	30·300	30·291	78·6	53·1	E	30	30·410	30·401	30·334	30·329	80·7	51·7	N

JULY, 1894.

KEW.								GLASGOW.							
	Barometer.				Temperature.				Barometer.				Temperature.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	30·261	30·233	30·171	30·159	80·7	58·2	E	1	30·331	30·279	30·190	30·153	75·6	52·9	ENE
2	30·114	30·105	30·057	30·080	77·9	59·2	SW	2	30·112	30·025	29·986	29·994	66·4	56·0	NE
3	30·057	30·102	30·094	30·143	69·2	54·8	NW	3	30·003	30·040	30·050	30·070	63·4	50·7	W
4	30·145	30·160	30·139	30·160	74·5	51·3	W	4	30·042	30·040	30·018	30·014	62·2	50·6	SW
5	30·159	30·146	30·078	30·074	77·2	52·0	S	5	29·999	30·003	30·005	30·004	69·1	57·1	S
6	30·014	29·958	29·868	29·873	83·4	58·4	SSE	6	29·944	29·920	29·738	29·735	72·1	55·9	SSE
7	29·908	29·967	29·975	30·022	69·8	53·3	SW	7	29·737	29·765	29·785	29·818	63·4	53·0	SW
8	30·041	30·051	30·007	29·941	70·0	49·8	SW	8	29·826	29·843	29·858	29·822	59·1	47·9	SW
9	29·805	29·773	29·724	29·718	71·2	55·8	W	9	29·748	29·701	29·659	29·617	62·3	52·9	W
10	29·653	29·575	29·437	29·299	65·3	55·3	SSW	10	29·548	29·501	29·440	29·384	61·2	52·7	W
11	29·278	29·354	29·387	29·399	67·5	53·1	W	11	29·329	29·296	29·274	29·256	66·6	48·6	WSW
12	29·364	29·309	29·479	29·610	66·0	52·3	W	12	29·227	29·221	29·235	29·291	65·1	48·6	W
13	29·617	29·626	29·591	29·644	66·7	53·9	SSW	13	29·294	29·342	29·391	29·471	62·7	53·0	SW
14	29·651	29·639	29·640	29·712	67·4	48·7	S	14	29·504	29·569	29·603	29·660	62·6	51·9	SW
15	29·739	29·844	29·881	29·947	62·4	51·9	W	15	29·682	29·765	29·786	29·824	64·1	51·9	W
16	29·952	29·911	29·796	29·773	62·2	53·5	SW	16	29·748	29·598	29·498	29·487	62·3	48·4	E
17	29·776	29·780	29·760	29·771	66·2	56·9	W	17	29·481	29·492	29·498	29·469	58·2	53·0	SW
18	29·655	29·639	29·635	29·695	65·1	54·6	W	18	29·430	29·482	29·521	29·572	66·4	51·2	W
19	29·695	29·740	29·803	29·875	65·2	53·6	WNW	19	29·605	29·655	29·696	29·741	65·1	46·3	W
20	29·889	29·920	29·908	29·889	66·3	53·2	SW	20	29·748	29·750	29·721	29·768	65·1	48·3	SSW
21	29·800	29·794	29·775	29·829	70·4	55·6	NW	21	29·769	29·728	29·707	29·748	61·9	47·4	WSW
22	29·851	29·871	29·842	29·808	67·7	54·0	W	22	29·778	29·819	29·823	29·886	64·1	50·0	W
23	29·731	29·761	29·824	29·889	61·2	56·8	N	23	29·920	29·956	29·972	30·070	67·1	45·2	E
24	29·924	29·957	29·914	29·913	73·3	58·0	NE	24	30·128	30·159	30·174	30·188	65·9	50·3	E
25	29·900	29·947	29·923	29·927	73·2	55·8	S	25	30·100	30·030	29·976	29·930	60·5	54·7	E
26	29·891	29·885	29·882	29·920	70·3	56·0	S	26	29·890	29·898	29·871	29·915	66·1	56·9	E
27	29·935	29·950	29·937	30·008	74·9	54·4	SW	27	29·923	30·048	29·941	30·013	69·9	57·8	E
28	30·044	30·062	30·039	30·051	74·3	53·5	SE	28	30·035	30·063	30·054	30·103	69·0	56·4	ENE
29	29·986	29·992	29·901	29·854	69·1	60·3	ENE	29	30·097	30·100	30·063	30·094	71·9	53·9	ENE
30	29·868	29·937	29·975	30·000	67·8	58·5	N	30	30·056	30·036	29·984	29·967	73·2	50·5	E
31	29·989	29·984	29·943	29·955	72·7	55·5	SW	31	29·943	29·908	29·866	29·832	69·9	54·0	NE

AUGUST, 1894.

1	29·903	29·861	29·798	29·805	68·3	58·9	SW	1	29·730	29·643	29·598	29·589	64·7	57·0	SW
2	29·736	29·637	29·575	29·539	67·3	57·9	SW	2	29·554	29·482	29·426	29·342	61·3	53·9	WNW
3	29·536	29·573	29·622	29·695	67·8	53·3	W	3	29·376	29·416	29·459	29·541	64·1	53·9	W
4	29·729	29·759	29·749	29·736	67·4	54·3	SW	4	29·558	29·572	29·536	29·584	60·9	53·2	SW
5	29·807	29·898	29·885	29·883	68·5	57·8	SW	5	29·628	29·638	29·605	29·609	60·2	49·4	SW
6	29·803	29·764	29·730	29·782	67·8	57·6	NNW	6	29·594	29·590	29·609	29·662	63·0	49·7	SW
7	29·811	29·867	29·879	29·882	69·3	56·3	SW	7	29·679	29·698	29·652	29·558	61·4	50·1	SW
8	29·793	29·769	29·685	29·663	67·8	58·1	SW	8	29·494	29·498	29·462	29·495	63·9	54·2	SW
9	29·687	29·763	29·803	29·859	66·9	54·9	W	9	29·522	29·600	29·653	29·735	66·4	52·9	W
10	29·833	29·897	29·931	30·037	66·2	54·7	WNW	10	29·790	29·953	29·979	30·029	63·9	52·7	W
11	30·054	30·097	30·101	30·131	64·3	52·5	NW	11	30·003	30·004	29·979	29·970	60·1	50·0	W
12	30·103	30·083	29·972	29·901	66·0	55·2	SW	12	29·867	29·799	29·690	29·723	60·3	54·5	SW
13	29·840	29·926	29·938	29·969	66·9	58·2	NW	13	29·783	29·838	29·865	29·786	60·8	51·0	W
14	29·855	29·757	29·633	22·613	75·7	57·6	SW	14	29·542	29·401	29·261	29·172	60·8	50·2	WSW
15	29·558	29·552	29·544	29·582	66·1	55·1	SW	15	29·116	29·176	29·265	29·461	62·2	50·0	W
16	29·624	29·719	29·827	29·952	61·1	51·7	NW	16	29·592	29·711	*	29·924	60·1	49·7	W
17	30·025	30·071	30·066	30·080	60·7	46·7	WNW	17	29·942	29·946	29·977	30·035	61·4	47·0	W
18	30·060	30·076	30·052	30·068	64·1	52·4	NW	18	30·012	30·013	30·007	30·005	61·4	49·9	W
19	30·045	30·028	29·952	29·864	62·4	51·4	SW	19	29·910	29·792	29·710	29 725	60·1	50·7	SW
20	29·807	29·849	29·849	29·905	64·4	53·4	WNW	20	29·705	29·766	29·814	29·835	60·0	47·1	WNW
21	29·885	29·858	29·755	29·714	61·0	45·5	SW	21	29·786	29·732	29·679	29·701	56·1	45·0	WNW
22	29·738	29·803	29·834	29·887	69·5	52·0	WSW	22	29·703	29·764	29·809	29·878	63·8	45·5	SW
23	29·856	29·792	29·767	29·849	59·2	56·7	E	23	29·900	29·950	29·969	30·062	63·4	44·7	NE
24	29·908	30·030	30·064	30·108	63·9	56·1	N	24	30·100	30·164	30·168	30·265	62·1	46·3	NE
25	30·052	30·072	30·025	29·977	60·3	56·6	E	25	30·281	30·262	30·202	30·196	59·4	44·8	E
26	29·905	29·940	29·936	29·983	73·0	56·2	SW	26	30·159	30·060	30·018	30·050	53·9	46·2	ENE
27	29·971	30·010	30·057	30·117	64·6	52·9	N	27	30·059	30·102	30·119	30·169	60·8	46·3	E
28	30·158	30·203	30·172	30·188	68·2	52·9	NNW	28	30·174	30·180	30·176	30·173	59·3	45·1	W
29	30·193	30·219	30·208	30·235	65·2	51·9	N	29	30·164	30·198	30·196	30·193	58·7	49·9	W
30	30·235	30·247	30·169	30·157	71·6	50·1	N	30	30·179	30·185	30·160	30·142	60·7	53·3	W
31	30·114	30·104	30·035	30·069	73·3	49·7	N	31	30·088	30·069	30·042	30·070	61·3	52·0	W

* Light out.

SEPTEMBER, 1894.

KEW.								GLASGOW.							
	Barometer.				Temperature.		Direction of wind at noon		Barometer.				Temperature.		Direction of wind at noon
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.		Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	
1	30·070	30·094	30·058	30·093	67·6	53·8	N	1	30·072	30·084	30·090	30·126	58·0	46·8	E
2	30·077	30·073	29·988	29·947	67·1	49·9	E	2	30·131	30·159	30·105	30·117	60·3	44·7	N
3	29·846	29·898	29·930	29·985	56·4	48·2	NE	3	30·088	30·082	30·023	30·059	58·1	40·9	W
4	29·997	30·047	30·026	30·075	60·9	45·3	N	4	30·068	30·096	30·085	30·120	59·1	46·7	NE
5	30·061	30·068	30·035	30·060	60·4	42·7	NW	5	30·120	30·155	30·138	30·191	59·2	49·3	N
6	30·044	30·078	30·064	30·123	57·5	42·2	N	6	30·186	30·198	30·157	30·138	57·3	44·6	S
7	30·105	30·102	30·035	30·018	59·3	44·2	W	7	30·062	30·034	29·971	29·972	58·1	43·0	W
8	29·881	29·835	29·887	29·976	58·9	48·4	N	8	29·960	30·002	30·063	30·181	58·8	41·1	N
9	30·026	30·131	30·195	30·314	61·8	50·6	N	9	30·259	30·366	30·384	30·420	60·3	40·1	SE
10	30·349	30·386	30·335	30·344	60·8	49·2	N	10	30·406	30·392	30·309	30·271	59·3	39·2	W
11	30·312	30·336	30·282	30·305	65·7	42·7	WNW	11	30·214	30·221	30·246	30·321	62·1	50·2	W
12	30·306	30·322	30·292	30·353	64·1	46·0	NNE	12	30·352	30·410	30·395	30·413	59·4	46·3	W
13	30·330	30·351	30·318	30·356	60·1	47·1	NE	13	30·414	30·421	30·361	30·391	64·4	44·9	W
14	30·332	30·356	30·321	30·351	62·4	51·9	NNE	14	30·382	30·391	30·368	30·386	60·8	47·0	W
15	30·337	30·359	30·323	30·338	60·3	51·4	NNW	15	30·386	30·414	30·390	30·407	62·8	48·1	W
16	30·306	30·320	30·284	30·308	59·8	52·9	NE	16	30·387	30·398	30·355	30·373	64·0	43·9	N
17	30·268	30·267	30·222	30·221	59·0	53·3	NE	17	30·360	30·369	30·328	30·374	61·7	45·9	ENE
18	30·157	30·139	30·074	30·081	64·0	51·6	ENE	18	30·344	30·335	30·266	30·260	56·3	50·3	E
19	30·057	30·087	30·057	30·086	66·0	51·3	E	19	30·209	30·202	30·179	30·186	56·0	50·1	ENE
20	30·069	30·079	30·021	30·035	61·1	52·3	NNE	20	30·170	30·178	30·136	30·135	57·3	51·0	ENE
21	29·998	29·988	29·941	29·925	57·0	54·1	NE	21	30·087	30·068	30·024	30·023	57·7	46·8	E
22	29·872	29·862	29·825	29·798	56·3	51·1	ENE	22	29·986	29·974	29·929	29·915	56·9	48·0	ENE
23	29·733	29·739	29·722	29·756	62·9	55·2	E	23	29·873	29·884	29·867	29·901	54·2	48·6	ENE
24	29·745	29·775	29·768	29·775	62·1	55·0	NE	24	29·908	29·931	29·921	29·936	56·4	44·5	E
25	29·673	29·634	29·572	29·606	59·7	55·8	E	25	29·890	29·876	29·803	29·822	57·7	45·9	NE
26	29·628	29·767	29·876	29·985	56·3	47·3	N	26	29·846	29·921	29·948	30·049	56·1	47·1	W
27	30·053	30·154	30·193	30·283	58·0	41·0	N	27	30·130	30·207	30·237	30·344	54·3	37·0	S
28	30·290	30·299	30·256	30·289	54·6	36·0	NNE	28	30·360	30·396	30·387	30·410	52·7	35·9	E
29	30·277	30·293	30·277	30·361	56·8	37·5	N	29	30·402	30·447	30·460	30·506	54·5	44·0	NE
30	30·377	30·422	30·429	30·463	58·0	43·4	NNE	30	30·504	30·530	30·509	30·510	50·8	43·8	E

OCTOBER, 1894.

1	30·438	30·479	30·433	30·450	57·3	42·6	ENE	1	30·466	30·460	30·436	30·427	54·5	43·3	S
2	30·410	30·425	30·339	30·323	57·6	49·5	NE	2	30·428	30·424	30·374	30·374	56·4	40·8	SW
3	30·256	30·239	30·181	30·197	57·1	50·1	NE	3	30·340	30·330	30·277	30·300	55·2	39·6	W
4	30·165	30·108	30·012	30·014	57·9	48·9	N	4	30·298	30·314	30·269	30·242	52·7	34·3	SW
5	29·989	30·011	29·980	30·000	57·2	51·3	N	5	30·200	30·199	30·162	30·178	53·1	48·7	E
6	29·940	29·907	29·878	29·916	57·1	50·4	NE	6	30·121	30·106	30·054	30·048	53·0	47·6	E
7	29·912	29·963	29·975	30·027	58·2	45·5	NE	7	30·016	30·040	30·027	30·054	54·9	48·1	E
8	30·038	30·076	30·059	30·103	53·0	42·1	NE	8	30·046	30·066	30·024	30·052	57·3	49·3	ENE
9	30·133	30·170	30·174	30·197	51·1	47·2	NE	9	30·043	30·037	29·986	29·930	57·3	47·9	SSW
10	30·160	30·141	30·119	30·144	59·0	50·2	S	10	29·924	30·009	30·049	30·130	59·6	47·2	W
11	30·154	30·215	30·214	30·266	61·7	55·9	N	11	30·158	30·219	30·206	30·240	55·0	39·4	N
12	30·270	30·305	30·260	30·273	61·0	45·6	E	12	30·202	30·198	30·176	30·174	61·0	47·7	SW
13	30·221	30·205	30·085	30·035	60·8	45·1	E	13	30·125	30·118	30·145	30·151	59·4	49·0	WSW
14	29·940	30·001	30·005	30·035	55·0	40·9	N	14	30·182	30·236	30·239	30·281	50·1	40·7	NE
15	30·043	30·087	30·077	30·115	51·0	39·1	NNW	15	30·274	30·314	30·224	30·305	49·6	40·1	N
16	30·109	30·122	30·105	30·153	50·2	36·9	NNW	16	30·278	30·283	30·241	30·222	49·9	32·8	WNW
17	30·123	30·096	29·998	29·949	50·7	32·6	NNW	17	30·151	30·061	29·969	29·910	49·3	38·5	WSW
18	29·841	29·777	29·673	29·642	48·2	39·5	NNW	18	29·861	29·861	29·800	29·792	44·7	37·1	E
19	29·613	29·640	29·606	29·600	48·8	39·8	NNE	19	29·758	29·795	29·774	29·798	41·9	29·7	NE
20	29·521	29·475	29·459	29·440	47·1	42·4	NNE	20	29·748	29·708	29·709	29·740	43·8	36·4	ENE
21	29·407	29·474	29·634	29·780	50·0	35·2	SW	21	29·693	29·707	29·764	29·811	46·2	34·8	E
22	29·785	29·779	29·782	29·873	48·7	33·8	NE	22	29·814	29·836	29·844	29·919	46·6	28·9	W
23	29·916	29·962	29·912	29·798	55·7	40·8	E	23	29·966	29·974	29·838	29·617	41·6	28·9	ENE
24	29·588	29·292	29·069	29·185	58·2	52·1	S	24	29·411	29·164	28·816	28·652	53·8	38·6	E
25	29·172	29·150	29·076	29·184	57·9	51·0	SW	25	28·652	28·696	28·707	28·968	52·4	47·5	W
26	29·356	29·516	29·404	29·244	59·2	50·6	S	26	29·317	29·461	29·399	29·271	47·8	38·7	ENE
27	29·196	29·249	29·295	29·431	56·1	49·7	SW	27	29·218	29·296	29·391	29·504	46·3	38·9	NE
28	29·480	29·635	29·689	29·639	55·4	47·2	SW	28	29·526	29·550	29·502	29·402	46·1	36·9	E
29	29·549	29·641	29·613	29·690	55·6	45·8	SW	29	29·272	29·308	29·314	29·400	47·6	40·3	SSW
30	29·714	29·752	29·821	29·885	52·2	48·2	SW	30	29·409	29·628	29·719	29·835	48·7	39·7	W
31	29·935	30·023	30·035	30·048	59·9	50·1	S	31	29·848	29·801	29·692	29·606	56·6	36·3	ENE

NOVEMBER, 1894.

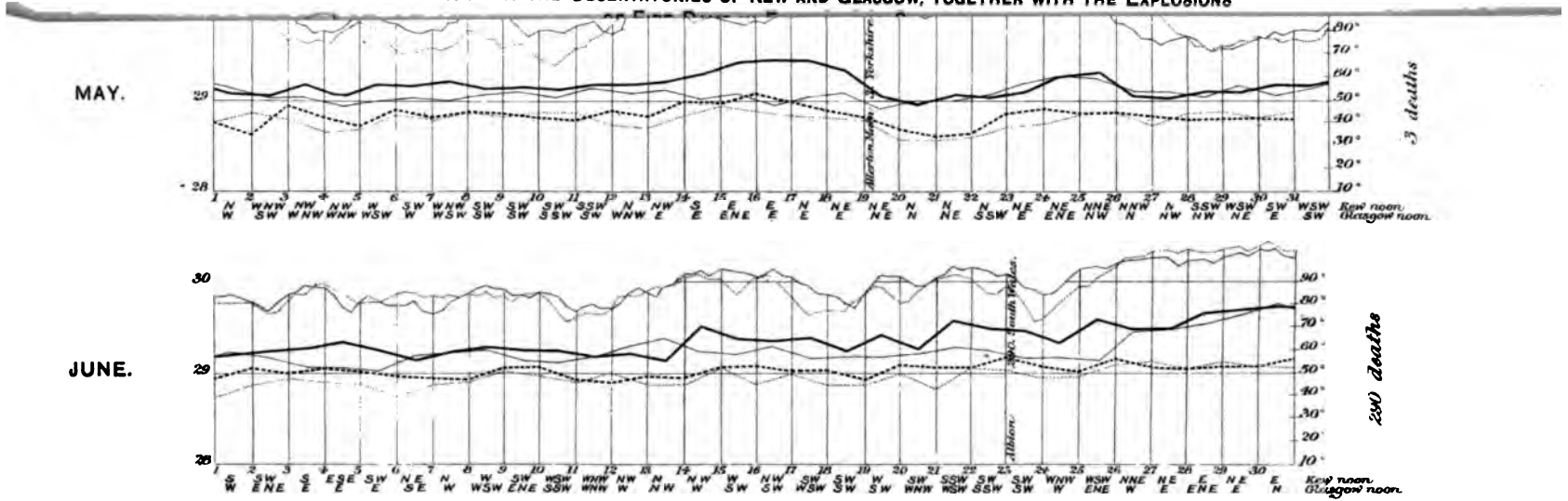
KEW.								GLASGOW.							
	Barometer.				Temperature.				Barometer.				Temperature.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	30·013	29·995	29·905	29·819	61·9	54·6	S	1	29·553	29·592	29·513	29·495	57·6	52·1	S
2	29·783	29·814	29·827	29·842	59·2	55·1	S	2	29·402	29·465	29·561	29·637	57·2	50·1	SW
3	29·791	29·749	29·714	29·726	61·0	53·1	S	3	29·582	29·482	29·396	29·367	55·8	49·5	SE
4	29·763	29·909	29·951	29·966	57·7	50 7	SW	4	29·435	29·591	29·622	29·536	51·0	47·2	SW
5	29·916	29·909	29·910	30·046	59·3	46·8	SSW	5	29·459	29·489	29·693	29·809	53·1	46·3	W
6	30·110	30·169	30·114	30·075	55·1	38·0	W	6	29·841	29·912	29·856	29·762	51·9	47·1	SW
7	29·960	29·841	29·589	29·403	57·0	39·2	S	7	29·589	29·438	29·299	29·279	52·1	45·1	S
8	29·505	29·625	29·677	29·722	51·1	42·0	W	8	29·294	29·380	29·377	29·486	47·8	41·3	WSW
9	29·715	29·689	29·594	29·570	52·7	41·7	S	9	29·512	29·424	29·330	29·367	51·1	45·1	S
10	29·544	29·559	29·498	29·415	55·1	44·1	W	10	29·308	29·300	29·242	29·201	48·1	41·9	SW
11	29·365	29·428	29·479	29·272	50·3	41·7	WSW	11	29·151	29·172	29·086	29·039	46·3	40·5	SW
12	29·201	29·111	28·869	29·020	54·5	42·8	S	12	28·980	29·019	29·103	29·252	46·8	42·4	WNW
13	29·439	29·712	29·752	29·593	49·1	42·7	W	13	29·410	29·492	29·286	28·870	46·2	36·9	SW
14	29·340	29·136	28·931	29·199	52·6	44·4	S	14	28·788	28·816	28·809	28·890	48·9	40·4	SSW
15	29·211	29·294	29·470	29·636	49·1	42·7	SW	15	28·974	29·074	29·162	29·280	44·9	38·3	SW
16	29·736	29·867	29·936	30·022	51·9	42·1	S	16	29·397	29·652	29·657	29·712	47·0	39·1	SSW
17	30·063	30·145	30·143	30·114	52·7	43·4	S	17	29·694	29·727	29·809	29·905	53·1	42·2	S
18	30·051	30·120	30·141	30·194	51·4	38·1	SSW	18	29·969	29·986	29·946	30·016	51·2	41·2	SE
19	30·192	30·226	30·243	30·279	52·9	35·7	S	19	30·028	30·054	30·006	29·932	49·6	36·3	S
20	30·253	30·204	30·152	30·178	53·3	37·9	S	20	29·842	29·744	29·926	30·055	51·3	40·2	WSW
21	30·323	30·439	30·433	30·428	47·1	42·7	NW	21	30·144	30·191	30·134	30·061	46·1	39·9	SSW
22	30·372	30·345	30·300	30·309	49·7	30·3	S	22	30·000	29·976	29·990	30·130	52·6	44·7	SSW
23	30·280	30·304	30·269	30·288	40·3	31·7	S	23	30·177	30·251	30·292	30·390	45·5	38·8	SW
24	30·288	30·319	30·287	30·304	49·2	35·3	NE	24	30·432	30·470	30·448	30·438	44·4	33·1	E
25	30·302	30·329	30·320	30·327	47·1	39·7	ENE	25	30·420	30·424	30·401	30·413	47·3	40·6	E
26	30·306	30·329	30·301	30·306	41·7	39·3	NNE	26	30·390	30·424	30·422	30·470	42·4	33·0	E
27	30·302	30·343	30·314	30·321	43·6	40·9	NE	27	30·454	30·451	30·361	30·315	41·9	29·3	E
28	30·242	30·363	30·340	30·351	42·9	33·9	NW	28	30·290	30·282	30·218	30·178	49·3	41·3	SW
29	30·292	30·281	30·243	30·292	48·3	34·0	NW	29	30·116	30·148	30·270	30·376	50·4	42·4	W
30	30·374	30·470	30·455	30·482	45·3	36·3	NNW	30	30·431	30·490	30·480	30·463	47·0	40·0	SW

DECEMBER, 1894.

1	30·468	30·494	30·434	30·438	39·6	28·6	N	1	30·438	30·448	30·410	30·401	44·9	39·8	W
2	30·426	30·419	30·304	30 244	44·0	31·5	N	2	30·358	30·338	...	30·213	43·6	35·8	SW
3	30·132	30·067	29·918	29·843	46·1	37·5	ENE	3	30·129	30·084	30·009	29·945	37·1	31·4	NE
4	29·736	29·722	29·690	29·721	39·0	32·9	E	4	29·868	29·844	29·789	29·788	40·7	31·5	E
5	29·761	29·842	29·871	29·944	41·5	38·5	NW	5	29·819	29·886	29·895	29·910	41·7	38·6	NE
6	29·977	30·015	29·986	29·955	39·2	31·2	NW	6	29·882	29·869	29·822	29·802	43·0	38·2	SW
7	29·859	29·848	29·771	29·724	48·1	34·1	SW	7	29·710	29·600	29·492	29·571	45·5	40·8	SE
8	29·723	29·864	29·987	30·085	47·0	32·5	NW	8	29·719	29·934	30·004	30·053	43·2	33·0	W
9	30·104	30·127	30 653	30·052	43·9	31·2	SE	9	30·010	29·984	29·867	29·681	43·4	32·9	E
10	30·048	30·085	30·100	30·136	47·6	43·2	S	10	29·676	29·758	29·796	29·877	51·1	41·7	S
11	30·123	30·138	30·073	30·025	48·1	41·2	S	11	29·869	29·801	29·807	29·830	51·1	43·6	S
12	29·966	30·058	30·151	30·195	50·6	41·9	W	12	29·767	29·841	29·788	29·752	50·5	38·1	SSW
13	30·155	30·166	30·121	30·116	50·8	43·8	SSW	13	29·655	29·563	29·542	29·679	54·1	50·0	SSW
14	30·116	30·127	30·021	29·824	51·6	47·1	SW	14	29·834	29·926	29·872	29·690	50·5	44·2	SW
15	29·850	29·975	30·104	30·202	48·2	39·3	W	15	29·583	29·749	29·922	29·910	46·4	37·9	W
16	30·063	29·960	30·103	30·214	51·1	38·5	WNW	16	29·849	30·052	30·130	30·125	42·1	34·4	NNW
17	30·191	30·104	29·959	29·888	50·1	37·8	S	17	29·924	29·719	29·684	29·556	46·7	32·3	NW
18	29·603	29·473	29·532	29·476	51·3	41·8	W	18	29·190	29·123	29·102	29·166	46·4	37·5	WSW
19	29·452	29·480	29·559	29·704	46·0	40·7	W	19	29·178	29·340	29·532	29·781	44·9	33·8	W
20	29·830	29·973	30·058	30·157	44·3	38·3	NW	20	29·946	30·102	30·112	30·041	41·1	36·5	WSW
21	30·158	30·125	29·962	29·788	50·0	32·3	SW	21	29·854	29·851	29·691	29·152	49·5	39·1	SW
22	29·439	29·299	29·601	29·939	50·3	40·7	W	22	28·270	28·738	29·522	29·811	50·6	37·6	WNW
23	30·120	30·224	30·189	30·121	45·9	36·4	SW	23	29·887	29·966	29·9 6	29·893	44·9	38·1	SW
24	30·063	30·106	30·163	30·268	49·0	45·7	WSW	24	29·937	30·013	30·087	30·220	45·1	42·3	ENE
25	30·348	30·459	30·506	30·547	47·7	44·3	WSW	25	30·284	30·358	30·377	30·402	50·2	42·7	SW
26	30·529	30·520	30·464	30·404	47·3	43·1	W	26	30·318	30·398	30·465	30·528	49·1	39·1	WNW
27	30·538	30·605	30·602	30·633	43·9	34·6	NW	27	30·524	30·586	30·622	30·588	43·9	37·0	W
28	30·586	30·481	30·142	29·751	45·1	32·0	SW	28	30·384	30·016	29·497	29·278	48·2	34·3	SW
29	29·496	29·374	29·346	29·375	46·7	36·3	W	29	28·986	29·041	29·184	29·247	39·1	32·3	WNW
30	29·319	29·333	29·373	29·488	38·1	32·2	NW	30	29·284	29·451	29·594	29·620	36·5	28·1	NE
31	29·540	29·640	29·687	29·767	35·7	27·2	NW	31	29·755	29·856	29·912	29·982	34·9	28·5	NW

The Federated Institution of Mining Engineers
Transactions 1894 5

DIAGRAM SHEWING THE HEIGHT OF THE BAROMETER, THE MAXIMA AND MINIMA TEMPERATURES AND THE DIRECTION OF THE WIND AT THE OBSERVATORIES OF KEW AND GLASGOW, TOGETHER WITH THE EXPLOSIONS



The firm top line is the Barometer reading at Kew, taken at 4 a.m., 10 a.m., 4 p.m., and 10 p.m.
The dotted top line is the Barometer reading at Glasgow, taken at 4 a.m., 10 a.m., 4 p.m., and 10 p.m.
The lower lines are the Maxima and Minima Temperatures at Kew and Glasgow, observed respectively at midnight and 9 p.m.
Kew lines are thick and Glasgow lines are thin.
The figures attached to the districts shew the Number of Deaths caused by the Explosions.

North of England Institute of Mining & Mechanical Engineers.
Transactions 1894-5

And.ᵂ Reid & Compᵞ Lᵗᵈ. Newcastle on Tyne.

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EXPLANATIONS.

The — at the beginning of a line denotes the repetition of a word; and in the case of Names, it includes both the Christian Name and the Surname.

Discussions are printed in *italics*.

The following contractions are used:—

C.—Chesterfield and Midland Counties Institution of Engineers.

M.—Midland Institute of Mining, Civil, and Mechanical Engineers.

S.—Mining Institute of Scotland.

N.E.—North of England Institute of Mining and Mechanical Engineers.

N.S.—North Staffordshire Institute of Mining and Mechanical Engineers.

S.S.—South Staffordshire and East Worcestershire Institute of Mining Engineers.

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